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CONTRACTOR REPORT ON MINERAL PROCESSING

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Arlington Technical Services

Background papers
for the OTA assessment

Marine Minerals: Exploring Our New Ocean Frontier

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1. Offshore Titanium Heavy Metal Placers: Processing and Related Considerations

Consulting and R & D in Extractive Metallurgy, Energy and Environmental Technologies

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OFFSHORE TITANIUM HEAVY MINERAL PLACERS: PROCESSING AND RELATED CONSIDERATIONS

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OFFSHORE TITANIUM HEAVY MINERAL PLACERS

A. Titanium and Associated Heavy Minerals

1. Salient features of the industry

A few features that are set forth more fully in the separate Commodity chapter are emphasized here to aid in the transition between topics.

Production of titanium heavy minerals (Ti H.M.) is driven primarily by demand for TiO_2 pigments. Titanium metal production accounts for only a small fraction of Ti H.M. demand in the U.S. and worldwide. Accordingly, it is not appropriate to invoke titanium metal's critical or strategic label in evaluating economic potential of offshore titanium and associated heavy minerals (Ti/assoc. H.M.). Titanium metal production will not be considered further in this chapter.

There are no major direct end-uses for Ti H.M., and the objective in processing is generally to convert them to higher TiO_2-content, hence, higher value form. Rutile and anatase pigments (not the mineral concentrates of the same designation) are manufactured as nearly pure TiO_2 powders having special physical and optical properties. Approximate ranges of TiO_2 content of common Ti H.M. sources are depicted in Figure 1. It is significant that the prices of titanium mineral sources, including titania slags, increase very strongly with TiO_2 content (Figure 2). Present prices of the end-members, TiO_2 pigments, are about 78¢ and 72¢ per lb for the more common rutile form and the anatase form, respectively (Lynd and Hough, 1986).

Combined U.S. demand for TiO_2 pigments is large, approaching 1 million tons annually (Greek, 1986). Supplying the feedstocks for the domestic industry utilizes the output of two titanium mineral sands mines, and a synthetic rutile plant, plus imports of Ti H.M. concentrates and titania slag. Superficially, there would appear to be no major marketing obstacle to introduction of a new source of Ti H.M., whether from an onshore or offshore deposit. However, the price-and-product profile is such that even pure ilmenite ($FeTiO_2$, 52.65% TiO_2 content) is not considered high grade in the present context.

Clearly, if titania is present in mineral sands as its major primary mineral, ilmenite, it is preferable that the ilmenite have been highly weathered to altered ilmenite and leucoxene. Our perception, based on these considerations and on the product mixes of mineral sands mining operations (Kaiser Engineers, 1982; Bureau of Mines, 1985; other sources) in Australia and the U.S., is that production of ilmenite alone is not currently economic. By-product or co-product recovery of more valuable mineral concentrates, especially rutile, monazite and the closely related xenotime (neither appearing to be abundant in known deposits in the U.S.), and zircon appears to be necessary for economic feasibility. Recovery of less valuable minerals (such as the silicates, kyanite, garnet and staurolite) may be feasible provided that quantities are sufficient, properties are suitable for the intended uses, and local markets exist for the concentrates. The overall economic feasibility would not normally be predicated on recovery of these lower value by-products.

In the remainder of this chapter, focus will be on the titanium minerals suite (ilmenite, altered ilmenite, leucoxene and rutile) and zircon.

2. Recent history of titanium sands mining in the U.S.

This subject is touched upon here mainly to provide some relevant background for extrapolation to offshore and for gauging possible commercial interest in titanium and associated heavy mineral deposits in the EEZ. Informative accounts of earlier heavy mineral sands mining, mainly in Florida but also over the border into Georgia, and in New Jersey, have been given by Pirkle et al. (1974), Lynd and Lafond (1975), Lynd and Hough (1980), and Garnar (1978), who has also provided a concise update (1985).

Briefly, a number of mines ceased operation because of depletion of reserves, including several along and not far inland from Florida's East Coast (northern half) and the Lakehurst, N.J. operation of SCM Corp., which shut down in 1978. The Folkston, Georgia heavy mineral deposit, discovered in 1952 and operated by Humphreys Mining Co. (formerly Humphreys Gold Corp.) for DuPont, closed in 1974. Mining and primary concentrating equipment were moved a few miles south to Boulougne, Florida. Reserves were again depleted, and the Boulougne operation shut down in 1979.

Presently, there are only two deposits producing heavy minerals in quantity in the U.S. Both are located in Northeastern Florida, namely, the Trail Ridge deposit of DuPont near the town of Starke and the Green Cove Springs deposit of Associated Minerals (USA), Inc. near the community of the same name. The old DuPont Florida Plant, as it is now designated, has been producing for over 30 years. Exploration in the early 80's effectively doubled known reserves at Green Cove Springs (Associated Minerals company brochure). It is a fair guess that both mines will be playing out in another two decades or so.

In addition to these major H.M. sands deposits, there are said to be smaller, nonproducing deposits in New Jersey, Tennessee, South Carolina, Georgia and Florida (Garnar, 1985). Limited identified reserves is reason enough for genuine interest,on the part of present producers and users, in potentially economic deposits of titanium heavy mineral sands in accessible areas of the EEZ.

There have also been a few failures, from which, lessons for offshore heavy minerals exploitation may be drawn. As recounted by Carpenter et al.(1953), a plant processing a H.M. sands deposit near Jacksonville, Florida in the early 1940's ran into difficulties owing to low capacity in mining and concentration and to the presence of organic slimes in the ore. The mineral processing scheme being used (by Rutile Mining Co. of Florida) combined wet concentrating tables and flotation. Humphreys Gold Corp. are said to have improved plant capacity and economics by introducing use of hydraulic dredge mining and their newly developed spiral concentrator.

The Manchester, N.J. mine of ASARCO, Inc. began operation in 1973 at 155,000 st/yr of ilmenite concentrate (63% TiO_2). Anticipation was for 20-22 years of production, with an increase to 165,000 st/yr (Lynd and Lefond, 1975). The operation closed down prematurely in 1982, for economic reasons. The deposit is also of marine placer origin. The principal Ti H.M. is 63%-TiO_2 ilmenite, with no appreciable rutile and leucoxene, but with small amounts of zircon and lesser amounts of monazite. There were, apparently, attempts at producing zircon and a zircon-rich intermediate but, through the life of the operation, ilmenite concentrate was the only commercial product.

This bears out our contention that production only of ilmenite from mineral sands is not likely to be economic, even for a deposit on land. High-grading in the early years meant mining 3.1% TiO_2 ore reserves; at the end, reserves being mined still contained a very respectable 2.6-2.7 % TiO_2. Design throughput was 1000 st/hr, and dredging was done at that rate (Crowell,1986). Owing to problems with slimes (note, again), actual dry mill production corresponded to a mining rate of about 900 short tons per hour (st/hr). Prospects for resuming operation or of relocating major equipment are remote at this time.

The Green Cove Springs mine of Titanium Enterprises (American Cyanamid and Union Camp Corp.) began operation in 1972 with production of ilmenite, rutile and zircon plus small amounts of leucoxene, staurolite and monazite (Lynd and Hough, 1980). In June 1978, the mine and wet mill were shut down for economic reasons. For a time, the dry mill continued to operate on stockpiled tailings. In April 1980, the operation was purchased by Associated Minerals Consolidated, Ltd. (AMC) for $12 million. AMC planned to use their experience in Australia to increase production to the levels shown in Table I. The mine first became profitable in 1983 at a more favorable, higher operating level. The improvements in plant and practice entailed appreciable further investment, but the total cost to AMC was much less than the presently estimated start-up cost.

In Section C, we shall attempt a rough economic comparison of onshore and offshore titanium mineral sands mining. An existing operation having the same production rate as the Table I case will serve as the onshore reference. Although 150,000 st/yr total H.M. concentrates is somewhat lower than world average for mineral sands operations, the products are sold, rather than mainly being used captively. Consequently, more information is publicly available with which to construct a production profile, as in Table I. It will be apparent from price lists for the individual concentrates (Table II) that, when markets for rutile and zircon are not depressed, these minerals each contribute more revenues than do the total ilmenite heavy minerals. Of course, their recovery also entails substantial production costs but, on balance, zircon could be considered at least an important by-product and rutile a co-product of the Green Cove Springs plant in normal markets.

B. General Extrapolations to Offshore Deposits

1. Typical flow sheets for processing Ti/assoc. H.M. sands

Heavy mineral sand placers have been subjected to natural mineral processing forces. The residual, chemically and mechanically durable mineral grains have been largely liberated, i.e., reduced in size and weathered sufficiently that individual grains predominantly consist of a single mineral, and classified, i.e., concentrated according to size, shape and density parameters. In the favorable circumstance where there has been no appreciable accumulation of overburden or infiltration of gangue materials, such as silt, clays, phosphorite, calcium carbonate, etc., and where induration by gangue materials has not occurred, mining and mineral processing to marketable concentrates are potentially relatively straightforward. However, achievement of efficient, economic operation of the mineral processing plant is by no means guaranteed, as has been noted.

Various types of separation equipment, connected in a number of possible series and parallel arrangements, are employed in mineral sands operations, the specific configurations adopted depending upon experience and, in some cases, local preference in addition, of course, upon specific features of the ore.

The mining and upgrading of titanium placer heavy minerals are similar in concept and practice to those operations applied to tin placers. Tin heavy mineral concentrates may, in fact, contain recoverable amounts of ilmenite, zircon and monazite, among other H.M.'s. As described in the separate chapter on Mining, the scheme most relevant to the offshore consists of mining by dredge, preliminary concentration of the collective H.M.'s in a floating "wet" mill, and separation and final concentration in a fixed "dry" mill.

Lynd and Lefond (1975 and 1983) give an excellent, detailed review of practices in the titanium beach sands industry. Figure 3, from their review, shows a generalized flowsheet for treating titanium beach sands. In the illustration, spirals are used as the principal concentration equipment in the wet mill. Recovered heavy minerals, in this example, are separate concentrates of ilmenite, rutile, zircon and monazite; the discarded minerals include quartz, plus epidote and other light heavy minerals. Other combinations of value minerals and gangue minerals are encountered in specific operations.

The major function of the wet mill is gravity (density) concentration, for which mineral processing operation, unconsolidated Ti H.M. beach sands are generally well suited. As remarked by Aplan, flowing-film types of gravity concentration equipment can be used to advantage here, including cones (operating on a modified sluicing principle) and spirals of various construction and features. In some plants, different types of separation equipment are used in combination (e.g., Fig. 4). Jigs have also been adapted for concentrating Ti H.M. sands (Lynd and Lefond, 1975), and their use for offshore H.M. placers is strongly supported by the dredge mining expert. Cross-sectional drawings and operating principles of a type of cone, a type of spiral and an assortment of jigs are shown in Fig. 5 a-c.

Humphreys spirals are the prevalent type of H.M. beach sands wet concentrator in the U.S., as illustrated by the accompanying flow sheet (Fig. 6) for the DuPont wet mill at Trail Ridge (Aplan, 1985). Note the intricacy of the circuit, involving sequential treatment of concentrate (C, tapped from upper turns of the spirals) and recycling of middlings (M, tapped from lower turns) and tailings (T). According to the figure, slurry feed (in the range of 20-22% solids) containing about 4% H.M. is concentrated to about 85% H.M. as dried feed to the dry mill. The function of the hammer mill circuit, treating the oversize from the initial screening, is to break up clumps of clay and mud for recovery of entrapped H.M.'s. Treatment of the H.M. concentrate (25-27% solids) with alkali (NaOH) was added to the flow sheet, as shown, for removal of an organic coating which had caused problems with separations in the dry mill.

Operation of the Trail Ridge dry mill, as of 33 years ago, was described by Carpenter et al. (1953). At that time, plant capacity was 112,000 st/yr titanium mineral concentrate, 45,000 st/yr staurolite, 30,000 st/yr zircon, and a smaller quantity of leucoxene. The published flow sheet is fairly detailed and difficult to read, and attention is directed to Fig. 3 with the comment that, at Trail Ridge, monazite is a very minor constituent of the ore. In the dry mill, approximately 800 st/day dried and heated feed from the wet mill is passed through high-tension electrostatic separators, where titanium minerals are separated from the silicate minerals. The titanium minerals are then separated by induced roll (high intensity) magnetic separators into a magnetic, 63%-TiO_2 ilmenite fraction and a nonmagnetic leucoxene fraction, which is cleaned electrostatically to about 80% TiO_2. There may not be sufficient rutile present to separate from this high-TiO_2 concentrate. The titanium mineral products are used internally by DuPont in TiO_2 pigment production.

Silicate reject from high tension separation is passed through induced roll magnets to recover staurolite, containing some tourmaline. The nonmagnetic fraction is treated to recover the zircon, in a separate plant. There, quartz and aluminum silicate minerals such as kyanite and sillimanite are removed by spiral concentration of the reslurried ilmenite-staurolite tailings. The further cleaned zircon concentrate is marketable at a much higher price than the staurolite.

2. Modifications for processing an offshore ore

It is anticipated that provisions would be retained in the wet concentration plant to remove organic material and coatings and to screen and break up or reject agglomerate material in the dredged sands. At sea, the sands slurry medium will, of necessity, be ocean water rather than aquifer or river water. Allowance for this, especially with respect to corrosion of equipment and piping, etc., is obviously indicated. The approximately 3% higher specific gravity of sea water as compared to fresh water is not expected to require major revision of construction or operation of separation equipment. There is ample experience with wet concentration of mineral sands using sea water as the slurry medium.

Washing of the H.M. primary concentrate to rid it of sea salts will be required in advance of the dry mill. The appropriate stage would be before dewatering and drying (Figure 6). As cautioned by R.J. Fantel (comments on draft report), there is not complete assurance that H.M. sands mined offshore will respond in the same way to mineral processing methods now employed for H.M. beach sands. This is a valid point and underscores the importance of including amenability testing for beneficiation and separation, with use of representative sample lots, at an early phase of offshore orebody assessment. As for any H.M. placer ore, a developmental effort will be required to optimize mineral processing equipment and procedures. Absent specific information from flowsheet development, it is reasonable to assume that the cost of the dry mill operation will depend mainly on total annual production, as for current Australian operations, and be less sensitive to the proportions of individual mineral constituents of the dry mill feed or products.

Much greater uncertainty is attached to costs and operability of a wet mill concentrator at sea. Motion compensation, whether of selected sections of the onboard plant or of the entire platform, is certain to be expensive. The alternative may be to accept a lower concentration efficiency, thereby losing a larger quantity of dredged value minerals in the tailings and probably also decreasing the grade of the concentrate. That means that, for the same production, dredging and wet-milling capacities and, hence, costs would be increased over those for achieving typical overall H.M. recoveries (about 70-80%).

A mitigating factor with respect to effect of motion at sea on concentration efficiency is that the large area requirement for wet concentration would make for a relatively stable platform. For example, at the Trail Ridge operation, the floating concentrator processing 22,400 st of feed per day, was carried on three connected barges with a total deck area of 27,000 sq ft (Carpenter et al., 1953). A more efficient, compact and less motion-sensitive type of concentrator may be needed for at-sea service than the spirals employed in that and other minerals sands operations. In any event, a good deal of development work will be required for this aspect of offshore Ti/assoc. H.M. production.

3. A dune sands analog of the postulated offshore deposit

Although the production of Ti/assoc. H.M. from offshore will be considered, for purposes of rough costing, to be tied in with existing production in Northeastern Florida, a fairly close analogy to our offshore deposit characterization and simplified processing scenario exists elsewhere. Reference is made to the dune sands mine and milling operation at Richards Bay, on the east coast of South Africa. As described by Lee and Poggi (1978), average thickness of the mineralized sands, on a clayey sandstone bedrock, is about 20 meters. The sand contains about 14% total H.M., of which about 40% are the Ti/assoc. H.M., ilmenite, rutile and zircon. In Section C, we shall assume 5, 10, and 15% total H.M. in an offshore deposit.

As further described by Lee and Poggi, the sand grains are well rounded and 95% <100 mesh, >200 mesh (about 75-150 microns). From average reserve grades of 5.4% ilmenite, 0.28% rutile and 0.65% zircon, three concentrates are produced, including 95.5%-TiO_2 rutile. The low grade, 47% TiO_2, of the ilmenite concentrate was an important factor in the decision to produce, by smelting, a high TiO_2 titania slag, suitable for pigment production by either the chloride or sulfate process, and a pig iron co-product. The small quantity of magnetite present is separated out, but the other H.M.'s (pyroxene, amphiboles, leucoxene and monazite) are not, making them diluents in this case.

Figure 7 gives a pictorial overview of the Richards Bay complex. The mining operation is large, about 20 x 10^6 tons annually, and two dredges are employed in the artificial pond. The floating concentrator separates H.M.'s by means of a complex circuit of Reichert cones and spiral concentrators. The primary concentrate is pumped directly to shore as a slurry, but otherwise the gross features of the operation are similar to those of the scenario for offshore.

It is worth noting that chromite presence in the dune sands is an expensive nuisance at Richards Bay, requiring intensive processing to remove it from the ilmenite concentrate in order to meet chromium specifications for pigment feedstock-grade slag.

For the offshore extrapolation, it would not be necessary in mining to deal with scrub trees and other vegetation, or buried residues thereof, a common problem with beach and dune sand deposits. Related to that, a supposed further advantage of offshore, that land reclamation would not be a major cost factor, might not be fully realized. Bottom recontouring may be required in some locations. Moreover, because of the favorable nature and volume of mineral sand tailings, reclamation onshore has been known to result, eventually, in production of more valuable land.

C. Offshore Titanium Heavy Minerals Production Scenario

1. Approach to a framework for evaluation

Inasmuch as hard resource information is not presently available on titanium and associated heavy minerals (Ti/assoc. H.M.) placers off the southeastern U.S. coastal states, the designation of resource specifics for costing purposes would be an entirely arbitrary exercise. In order to make a connection, however tenuous, with actuality and in view of the very small project time allotment for the evaluation, the following framework was agreed upon.

a. The Ti/assoc. H.M. deposit is considered to be located in the Minerals Management Service exploration permit area in Federal waters offshore Georgia, as identified in MMS Atlantic OCS Region special information bulletin SI 2-86, 22 January 1986.
b. The best-case assumptions are made that the deposit consists of entirely unconsolidated heavy mineral sands without overlying sediments (overburden), having a total heavy mineral content of 5, 10 or 15%. Other deposit characteristics having particular relevance to the mining operation are specified in that chapter.
c. As a consequence of the assumed nature of the offshore deposit, it is considered that separation and recovery of individual heavy minerals can be effected by conventional mineral processing, as practiced (with variations) at numerous onshore H.M. sands operations.
d. It is assumed, rather arbitrarily, that the H.M. primary concentrate from the onboard wet mill will be processed at an existing onshore dry mill. Present or potential producers have other options available to them, but economic feasibility is not changed in a major way.
e. Consistent with the foregoing assumption, the annual production of the dry mill is taken to be 150,000 short tons (136,000 tonnes) of Ti/assoc H.M. An assumed rate of over 200,000 st/yr could equally well have been taken, but less information is available on the corresponding product mix.

Some additional assumptions are needed in order to determine mining and wet-milling rates for the three postulated ore grades and also the feed rate of primary concentrate to the onshore dry mill concentrator. The throughput of the dry mill is arranged to be the same in each case, namely, 150,000 st/yr. In rough correspondence with published information on present and former heavy minerals sands operations of northeastern Florida and adjacent Georgia, it is considered that:

f. Recoverable titanium and associated heavy minerals constitute 45% of total recoverable heavy minerals.
g. The overall recovery of H.M., including the Ti/assoc. H.M., is 70%; this figure is comprised of an assumed 80% recovery of H.M. in the floating wet mill concentrator and 85-90% (87½% avg.) in the dry mill (land plant).
h. Unseparated lighter materials, presumably mainly quartz sand, constitute 15% of the feed to the dry mill.

From the foregoing quantities, the mining rates are estimated as:

$$\frac{150{,}000 \text{ st/yr}}{0.45 \times 0.7 \times (0.05,\ 0.10,\text{ or } 0.15)} = \begin{matrix} 9.52 \times 10^6 \text{ st/yr(dry basis)} \\ 4.76 \times 10^6 \text{ st/yr(dry basis)} \\ 3.17 \times 10^6 \text{ st/yr(dry basis)} \end{matrix}$$

The estimated feed rate to the dry mill, which is the same as the throughput of primary concentrate in the wet mill, is:

$$\frac{150{,}000 \text{ st/yr}}{0.45 \times 0.875 \times 0.85} = 450{,}000 \text{ st/yr (dry basis)}$$

The assumed 80% recovery of H.M. in the onboard wet mill may be optimistic for the untested situation of operating in unprotected waters. Poorer performance of the gravity separation equipment would necessitate a higher mining rate and production of a larger quantity of lower grade primary concentrate. However, the assumed optimistic recovery figure is consistent with the "best-case" nature of this preliminary evaluation.

We have no special insights into the intent of present or potential producers with respect to exploitation of the offshore resource. One or the other might opt to construct a new dry mill near a convenient port, rather than use an existing plant located a much greater distance from the ore body. Alternatively, a present producer might opt to use his depreciated plant to process another's offshore concentrate on a toll basis. The main value of the scenario determined upon is that it illuminates many of the important factors in the consideration of offshore H.M. placer mining and processing. As a particular objective, it provides a framework for visualizing roughly how good an offshore deposit has to be, in terms of content of higher value heavy minerals such as rutile and zircon, in order to meet revenue requirements for break-even operation and for return on capital invested.

Briefly, the procedure adopted was to obtain component capital and operating costs for a conventional 150,000 st/yr Ti/assoc. H.M. sands operation. With that basis, the aggregate costs of mining, primary concentration and transport of the heavy mineral concentrate, as provided by the Mining chapter authors, could be compared to the corresponding costs for entirely shore-based production. The incremental costs for the offshore deposit could then be used to estimate the content of higher-value H.M. required to generate offsetting revenues.

An essential simplifying assumption is made at this stage, specifically, that the capital and operating costs of the dry mill concentrator are determined primarily by its production and capacity with respect to total marketable output of separated H.M. concentrates. In other words, costs of the dry mill operation are not sensitive to the proportions of individual heavy minerals in the feed. This appears to be the case for the numerous Australian beach sand operations, which provided the data base for the economic evaluation of titanium mineral availability (U.S. Bureau of Mines IC 9061, 1985, and Kaiser Engineers, 1982). Further refinements are not warranted at the present level of estimation.

2. Component costs for H.M. sands mining and processing

Dredging and milling costs for beach sands operations in Australia are reported in IC 9061 in January 1984 U.S. dollars.

Capacities were reckoned on an 82% operating factor, i.e., 300 days or 7200 hours of operation per year. The IC 9061 authors' reported costs were updated and adjusted, using equipment costs indices, to current dollars. Fixed capital requirements, estimated for each of the assumed ore grades and corresponding annual mining rates and for the common, approximately 450,000 st/yr (dry basis) primary concentrate, are summarized in Table III. Construction of a dredge and modification of the wet mill for operation at sea would increase those fixed capital requirements substantially, whereas dry mill costs would not, in the present scenario, be affected perceptibly.

Direct operating costs reported in IC 9061 were broken down into their major components, viz., operating labor, fuels and energy, and maintenance and repair costs. The components of cost were then updated to current dollars with use of Southeastern U.S. unit costs for each category. Estimated direct operating costs are summarized in Table IV for each assumed ore grade. The cost-updating methodology is illustrated in the Appendix.

Direct and indirect operating costs for dredging and wet milling at sea will be higher than for the land-based cases summarized above. Unit costs for labor and maintenance, for example, will be higher because of shipboard operating practices (e.g., 12-hour shifts) and the more corrosive environment, among other factors. Power will be more expensive because it must be generated on site in relatively small diesel electric plants, rather than being obtained from a central power station. Estimated labor and power requirements for land-based operations at each assumed ore grade are summarized in Table VI.

3. Ore grades for break-even production from offshore

Capital requirements for dredging and wet-milling at sea and transporting 450,000 st/yr (dry weight) of H.M. concentrate to shore have been estimated to be 74, 54, and 46 million dollars for the assumed 5%, 10%, and 15% ore grades, respectively (costs from Table 5-1 of the Mining chapter). These capital cost estimates are much higher, by factors of roughly 5 to 7, than the combined fixed capital costs for dredge and wet mill for the corresponding land-based operations (Table III). A factor in the differences, although certainly not the major factor, is the shorter operating time at sea: 225 days per year (5400 hours) on station at the mine site vs. 300 days (7200 hours) on-stream assumed in the titanium availability assessment (IC 9061). The resulting greater instantaneous throughputs translate into greater capacities and costs. The probably higher actual operating factors at the existing Florida plants do not change the picture in a major way at the present level of estimation.

From the above figures, fixed capital requirements for the production of primary concentrate are, respectively, \$165, \$120 and \$100 per annual ton. From Table 5-2 of the Mining chapter, break-even revenue requirements are estimated to be \$66, \$50 and \$44 per ton of primary concentrate, for 5%, 10%, and 15% ore grades. Even if these estimates are very conservative, closer estimates would clearly exceed the value of a wet mill concentrate containing ilmenite as the only economically recoverable mineral. Current U.S. East Coast price of ilmenite concentrate is \$45-50 per (short) ton (Lynd and Hough, 1986).

The breakdown of estimated capital requirements and operating costs for the production of 150,000 st/yr of product concentrates is given in Table V. This breakdown is based on depreciation schedules in use under current tax law and on the assumption that equity funding is used to finance the venture. Break-even revenues are seen to range from about \$250 to \$170 per ton of marketable products.

Given the risks inherent in developing an offshore deposit, the developers would expect higher returns than for a conventional mineral sands operation and require a more rapid payback. Under current tax law, a 3-year payback, for example, would require revenues of approximately \$420, \$315, and \$280 per ton of products for the 5%, 10%, and 15% ore grades, respectively. Clearly, unless the cost estimates for dredging, wet milling and transport to shore are grossly overblown, the Ti/assoc. H.M. suite would need to contain a high proportion of rutile (ca. \$355 per ton concentrate, f.o.b. U.S. east coast) or high-TiO_2 leucoxene (ca. U.S. \$230 per ton for min. 91% TiO_2, max. 1% ZrO_2 grade, f.o.b. West Australia). Presence of monazite (see Table II for price) in significant quantity is considered unlikely.

The market could probably accommodate 150,000 st/yr rutile from offshore mining, assuming that it would be competitively priced. The potential for accommodating a large new supply of leucoxene is less assured. The value of zircon concentrate (ca. \$145 per ton) is less than or comparable to break-even cost, so that a high-zircon offshore deposit of the H.M. grades assumed would probably also not be economically feasible to exploit under the general plan described for mining and processing.

Comparison can be made with an onshore mineral sands operation for the 5% ore grade case, for which component data are directly extractable from the titanium availability assessment (IC 9061). Thus, capital requirements, including mine development costs, to produce wet mill concentrate are about \$130 per annual ton of products. Cash operating costs to produce wet mill concentrate are about \$34 per ton of products. These figures are approximately 25-40% and 30-40%, respectively of the corresponding costs for offshore mining and primary concentration (Table IV).

Addition of dry-milling costs, working capital and general expenses increases capital requirements to about \$210 per annual ton and cash costs to about \$58 per ton of products for the land-based mine and plant. The break-even average price for the products would be about \$90 per ton. The same assumed 3-year payback would be obtained with product revenues of about \$165 per ton. This figure, for a 5% H.M. beach or dune sand deposit, is about half that for a 10% offshore deposit. While revenues from ilmenite alone would also not be sufficient for the onshore example, there is certainly greater flexibility here with respect to product mix and ore cut-off grade.

Alternate arrangements could be costed, in addition to that assumed for this preliminary assessment. Rather than use an existing dry mill, for example, a producer might elect to construct a new mill at an especially convenient location relative to the mine site; or, as mentioned earlier, the primary concentrate from offshore might be dry-milled on a toll basis; a different financing method might be elected, etc. Moreover, a number of cost items such as provision of port facilities, land transportation to the dry mill, and so forth, have not been accounted for explicitly here. However, their explicit inclusion and limited revision of the scenario, as indicated, are not expected to change the sense of the above rough estimates of total revenue requirements.

4. Ore grade requirements: revised basis and approach

Figures employed in Sections C1 and C2 for orebody compositional characteristics and production rates are substantially all derived from old or incomplete published information dealing with one or another Ti/assoc. H.M. sands operation in the southeastern U.S. Processing costs were obtained from the previously cited analysis (IC 9061) of Australian beach sands operations. In view of this pieced-together and not fully consistent evaluation basis, Associated Minerals (USA), Inc. kindly constructed and made available, for this revised section, a self-consistent set of H.M. concentrates production data and associated costs that can be considered more or less typical of H.M. sands mining in the southeastern U.S. The numbers and explanatory notes, largely as received from Richard Beale at the September 18 Workshop, are presented in Table VII (see p. E-4, Table VI, for explanatory notes). We are grateful for this unexpected assistance.

It will be noted that 254,000 (short) tons of primary concentrate containing 15% non-heavy minerals is proposed as a more representative annual production. From this primary concentrate, 145,000 tons (total) of separate H.M. concentrates is typically produced in a sequence of processing operations qualitatively similar to flow diagrams described above. For the production profile illustrated in Table VII and for the indicated prices of rutile, zircon, ilmenite and leucoxene concentrates, sales revenues would be $17.95 million vs representative total operating costs of $15.5 million. The calculated pre-tax profit of $2.45 million amounts to 13.65% of sales, or 15.8% of costs.

In this example, the orebody would be considered a rutile deposit with co-product or major by-product (depending upon one's definition) values of zircon and of ilmenite + leucoxene. Costs of mining and primary concentration, including of those related functions identified in Table VII, are shown as $7 million, 45% of total operating costs. By making a rather arbitrary, proportional assignment of administrative costs and annualized working capital to mining costs, as broadly defined here, and to processing costs, we arrive at the following rough estimates for a typical southeastern U.S. onshore H.M. sands operation:

Typical onshore costs in 1986 U.S. dollars

Sector	per ton ore	per ton concentrate	per ton product
mining and primary concentration	$1.0	$35	$62
processing to products	$0.7	$26	$45

If the Ti/assoc. H.M. deposit is visualized to be located in the offshore, costs through production of primary concentrate would be expected to rise sharply, as discussed previously. By way of indirect confirmation, the average cost (not price) of sand or gravel dredged offshore and brought ashore is about $4 per ton for established operations in the U.K. and Japan. Approximately the same cost has been indicated to apply to offshore sand and gravel mined along the Atlantic seaboard or at a dredge site served by the Port of Los Angeles (Huhta, 1986). This number is four times the above estimated costs, per ton of ore, of dredge mining and primary concentration of onshore H.M. sand placers.

The approximately $4 per landed ton of sand and gravel might be considered a representative cost for conditions that are neither exceptionally favorable nor more adverse than average. Costs mentioned at the Sand & Gravel workshop reported on by Huhta (1986) ranged from under $1 (credible?) to over $20 per ton. Washing and sizing, on land, of the dredged offshore sand and gravel add about $1 per ton to the cost.

By fixing selected compositional characteristics of the offshore placer deposit, one can estimate changed grade requirements as mining costs are increased by increment over those typical of onshore titanium mineral sands mining. For specificity, consider Ti/assoc. H.M. to remain the same 66% of the H.M. suite, and assume that the ratios of rutile to zircon and of ilmenite to leucoxene are maintained. In addition, assume that the H.M. grade remains the same at 3% and that the primary concentrate from offshore is amenable to processing in the same onshore plant and/or for the same cost. A corollary assumption is that recoveries do not change over those given or implicit in Table VII.

With the foregoing assumptions and simplifications, the increases in revenues required to keep profit at the same percentage of costs (or revenues), as mining and related costs are increased, can be estimated from the expression, $\triangle\text{Revenue} = \text{Factor x } (\$_{r,z} - \$_{i,l})z$. In this expression, $\$_{r,z}$ and $\$_{i,l}$ are the weighted, mean realized prices of rutile plus zircon and of ilmenite plus leucoxene, respectively, and z is the incremental fraction of rutile plus zircon necessary to yield offsetting revenues as here listed:

Mining Costs	Total Operating Costs	Profit	Profit as % of		Required Revenues
			Sales	Costs	
$\$\ 7\text{x}10^6$	$\$15.5\text{x}10^6$	$\$2.45\text{x}10^6$	13.6_5	15.8_1	$\$19.95\text{x}10^6$
\$14 "	\$22.5 "	\$3.56 "	same	same	\$26.06 "
\$21 "	\$29.5 "	\$4.66 "	same	same	\$34.16 "
\$28 "	\$36.5 "	\$5.77 "	same	same	\$42.27 "

The numerical Factor is calculated from the base case (mining costs = $7 million), and the incremental fractions, z, of rutile + zircon are estimated as follows:

Mining Costs		z	fraction of rut + zir	fraction of ilm + leu	%*
Multiple	Amount				
1	$\$ 7x10^{6}$	0.0	0.24	0.42	36
X 2	$14 "	0.158	0.40	0.26	61
X 3	$21 "	0.315	0.55_5	0.10_5	84
X 4	$28 "	0.473	(0.71)	(-0.05)	(108)

---

*combined rutile + zircon as percentage of total Ti/assoc. H.M.

From estimates and evidence presented earlier, it is difficult to perceive how mining and primary concentration costs could be less than doubled by moving those operations offshore. If those costs would be trebled or more, results of the above simplified analysis suggest that rutile and zircon would need to be present in considerable excess over ilmenite plus leucoxene, at the same assumed 3% total H.M. (of which Ti/assoc. H.M. were assumed to comprise 66%). The broad conclusion here is consistent with that arrived at under a different set of assumptions and postulates in previous sections. Restated, the conclusion is that the production of offshore placers primarily for low-value minerals such as ilmenite will not be economically feasible, unless means are found to deliver primary concentrate to the onshore processing plant at costs comparable to those incurred in producing the equivalent onshore placer.

A possible direction for reducing costs for an offshore H.M. placer deposit would be to adopt dredge mining and transport methods that have evolved and become well established for offshore sand and gravel. If sufficient revenues could be realized from the sale of tailings to offset the risk-weighted cost of the primary concentrate as delivered to the processing plant (referred to earlier as the dry mill concentrator) on land, a comparable ore grade could then be of commercial interest.

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Table I . Recent production history at Green Cove Springs, Florida (thousand metric tons per year), Associated Minerals (USA)

	Planned (Lynd and Hough, 1980)	Company Brochure	Industrial Minerals June 1986 issue
ilmenite	45	50	63 (63+% TiO_2)
rutile	22	ca. 30	25-27 (95% TiO_2)
zircon	22	ca. 35	35
leucoxene + altered ilm.		smaller quantities	12 (65% TiO_2)
monazite			
staurolite			
			136,000 tonnes/yr
Titanium and associated H.M. production:			150,000 st/yr

Table II Market prices of titanium concentrates and related minerals for January 1984 (*22-24*)

Commodity	Where applicable (f.o.b.)	Grade, pct	Price, $/mt
Titanium product:			
Ilmenite concentrate	Mill, Australia	[1]54 + TiO_2	$32.00
Do	Mill, United States	[1]54 + TiO_2	42.00
Leucoxene concentrate.	Mill, Western Australia	87 TiO_2	225.00
Rutile concentrate	Mill	95 TiO_2	347.00
Synthetic rutile	Plant, Mobile, AL, United States	90 + TiO_2	350.00
Titanium slag	Sorel, Quebec, Canada	71 TiO_2	159.00
Do	Richards Bay, Republic of South Africa	85 TiO_2	181.00
Byproduct:			
Garnet	Mill	Abrasive	10.00
Magnetite	..do	NAp	23.00
Monazite concentrate	..do	55 REO	389.00
Pig iron	..do	NAp	235.66
Zircon concentrate	Mill, Australia	65 ZrO_2	104.50
Do	Mill, United States	65 ZrO_2	182.00

[1] Price does vary on TiO_2 grade (from ~47 to ~64 pct TiO_2). This also applies to the values listed in the text.

From Fantel, Buckingham and Sullivan (1986)

Table III. Estimated fixed capital costs for production of 150,000 st/yr H.M. concentrates from an onshore deposit for three ore grades.

Ore Grade, Total H.M. Content†:	5%	10%	15%
Capital Requirements, $million			
Dredge	1.6	0.6	0.4
Wet Mill	14.0	7.4	5.8
Dry Mill	4.4	4.4	4.4
Total Fixed Capital	20.0	12.4	10.6

---

† Of which, Ti/assoc. H.M. = 45% by wt.

Table IV. Estimated operating costs for production of 150,000 st/yr H.M. concentrates from an onshore deposit for three ore grades.

Ore Grade, Total H.M. Content+:	5%	10%	15%
Operating Costs, $million/year			
Dredge Cash Costs	1.2	0.7	0.6
Wet Mill Cash Costs	3.9	3.1	2.9
Dry Mill Cash Costs	1.8	1.8	1.8
General and Admin. Expenses*	1.9	0.8	0.6
Total Cash Costs	8.8	6.4	5.9

+ Of which, Ti/assoc. H.M. = 45% by wt.

* Combined costs of general production overhead, services, water, provision of stores, and laboratory operations.

Table V . Rough cost breakdown for production of 150,000 st/yr H.M. concentrates from an offshore deposit at three specified ore grades.

Ore Grade, Total H.M. Content†:	5%	10%	15%
Capital Requirements, $million			
Dredge*	40	36	32
Onboard Wet Mill*	34	18	14
Onshore Dry Mill	4.4	4.4	4.4
Total Fixed Capital	78.4	58.4	50.4
Working Capital	7.8	5.8	5.0
Total Investment	86.2	64.2	55.4
$/annual ton products	575	430	370
Operating Costs, $million/year			
Dredge plus Wet Mill Cash Costs	17.6	13.8	12.7
Dry Mill Cash Costs	1.8	1.8	1.8
General and Admin. Expenses	1.9	0.8	0.6
Total Cash Costs	21.3	16.4	15.1
Depreciation Expense	15.7	11.7	10.1
Total Operating Cost	37.0	28.1	25.2
$/ton products	245	185	170

---

† Of which, Ti/assoc. H.M. = 45% by wt.

* From Table 5-1 of the Mining chapter (Richardson and Horton).

Table VI. Estimated labor and power requirements for land-based mineral sands operations producing 150,000 st/yr H.M. concentrates.

	Ore Grade, %H.M.		
	5%	10%	15%
Dredge			
Operating staff	7	4	3
Maintenance staff	6	4	3
Power requirements, MW	1.39	0.85	0.69
Wet Mill			
Operating staff	73	59	54
Maintenance staff	13	11	10
Power req'd, MW	2.05	1.65	1.52
Dry Mill			
Operating staff	40	40	40
Maintenance staff	4	4	4
Power req'd, MW	0.50	0.50	0.50

Explanatory notes for Table VII (next page):

Typical mining operation consists of:

suction dredge and concentrator with wet gravity separations, land clearing/restoration, and field services (concentrate handling, etc.).

Typical processing operation consists of:

attritioning, drying, high-tension (electrostatic) and magnetic separation, wet gravity separations, and shipping.

Administration consists of:

stores, laboratory facilities and administrative support.

Included in typical costs of operation (Table VII) are:

depreciation, engineering, royalties, overhead, sales, etc.

Table VII. Representative production profile and costs for an onshore heavy mineral orebody in the southeastern United States.

Ore Grade

H.M. content of 3.0%, of which 9% rutile, 15% zircon, 40% ilmenite and 2% leucoxene (i.e., 66% Ti/assoc. H.M. and 34% other H.M.)

Mining (9 x 10^6 short tons per year)

1200 tons per hour, 85% running time, 80% average H.M. recovery in primary concentrator, 85% H.M. grade of primary concentrate:

$$\frac{1200 \times 7500 \times .03 \times .80}{.85} = 254{,}000 \text{ tons concentrate}$$

For reasonable average recovery figures for rutile, zircon, ilmenite and leucoxene, cumulative primary concentrate would contain:

rutile	- 23,100 tons
zircon	- 37,300 "
ilmenite	- 86,400 "
leucoxene	- 4,300 "

Processing

Assuming that all primary concentrate produced by the mining operation is consumed by the processing plant, the products are:

rutile concentrate	: 26,500 tons	x $\$355$/ton	= $\$9.4 \times 10^6$
zircon concentrate	: 29,800 tons	x $\$150$/ton	= $\$4.5 \times 10^6$
ilmenite concentrate	: 84,700 tons	x $\$45$/ton	= $\$3.8 \times 10^6$
leucoxene concentrate	: 4,200 tons	x $\$60$/ton	= $\$.25 \times 10^6$

Total revenues = $\$17.9_5 \times 10^6$

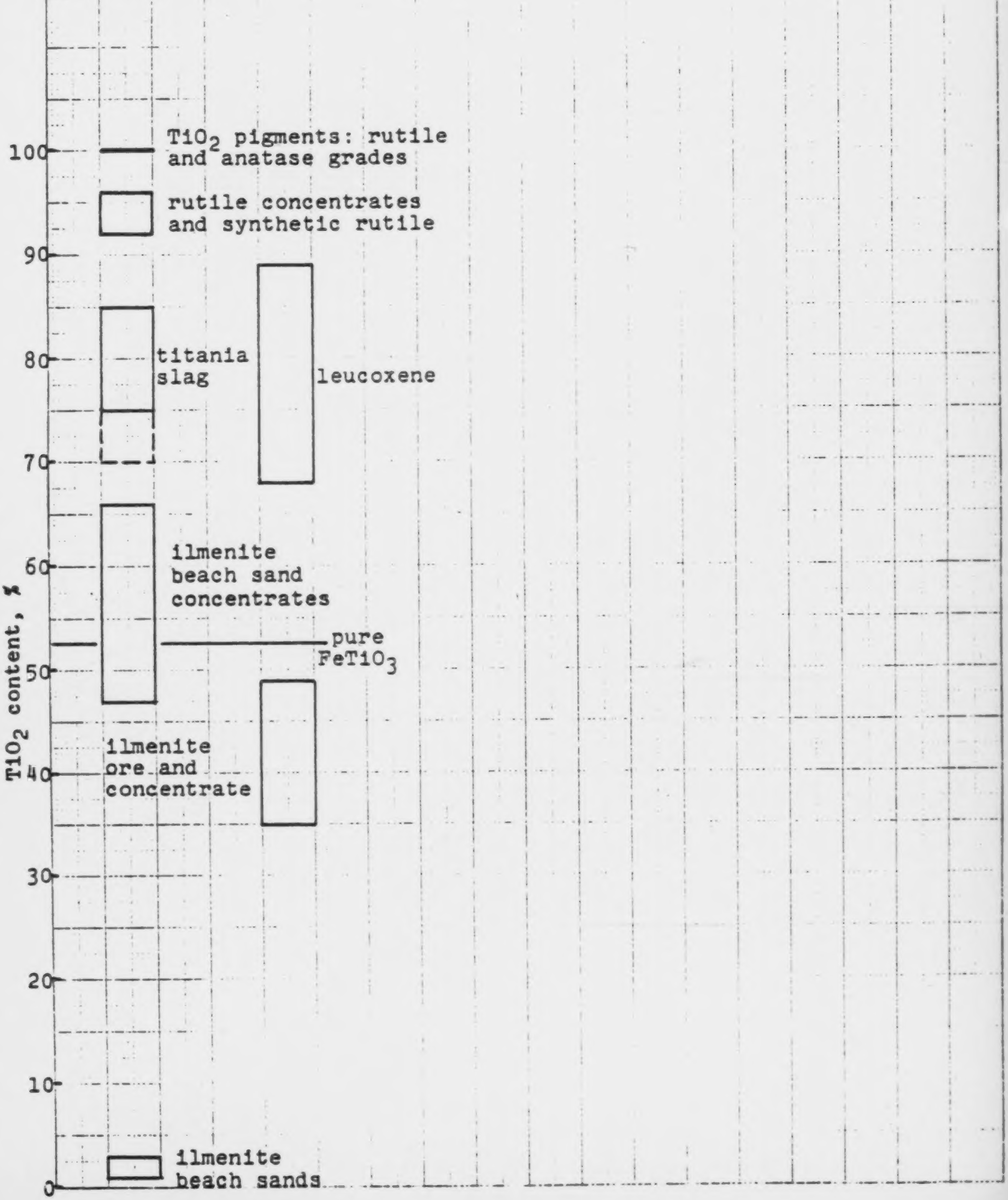
Typical costs of operation

mining	$\$7 \times 10^6$ per year
processing	$\$5 \times 10^6$ per year
administration	$\$1.5 \times 10^6$ per year
working capital	$\$2 \times 10^6$ per year
	$\$15.5 \times 10^6$ per year

Revenues	-	Costs	=	Profit
$\$17.95 \times 10^6$		$\$15.5 \times 10^6$		$\$2.45 \times 10^6$

See Page E-4 for explanatory notes

Fig.1. TiO_2 contents of major titanium mineral raw materials and intermediates.



10 X 10 TO THE CENTIMETER 46 1521
KEUFFEL & ESSER CO.

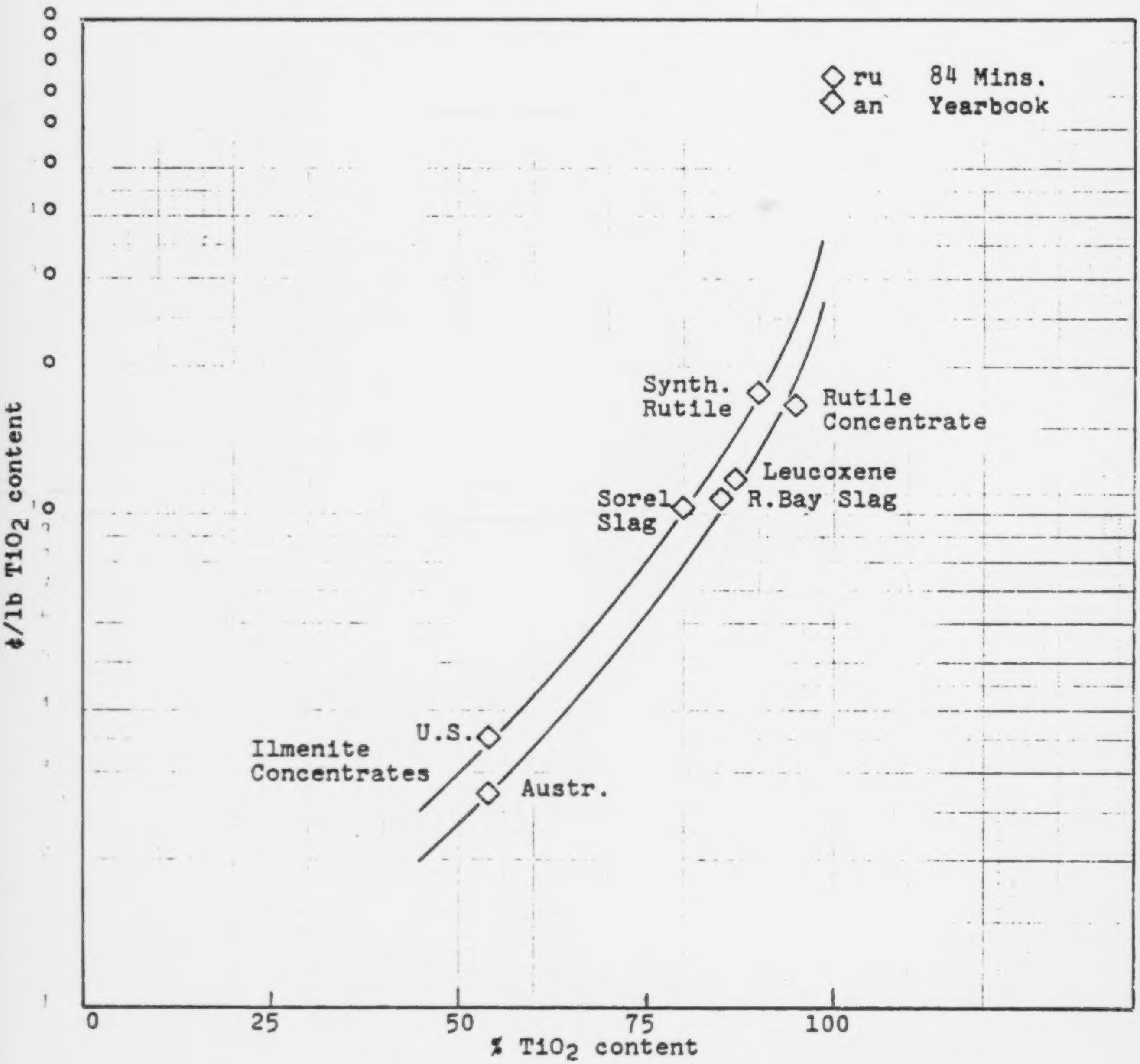


Fig. 2. Values of the TiO$_2$ content of common titanium mineral concentrates and intermediates (cf. Table II).

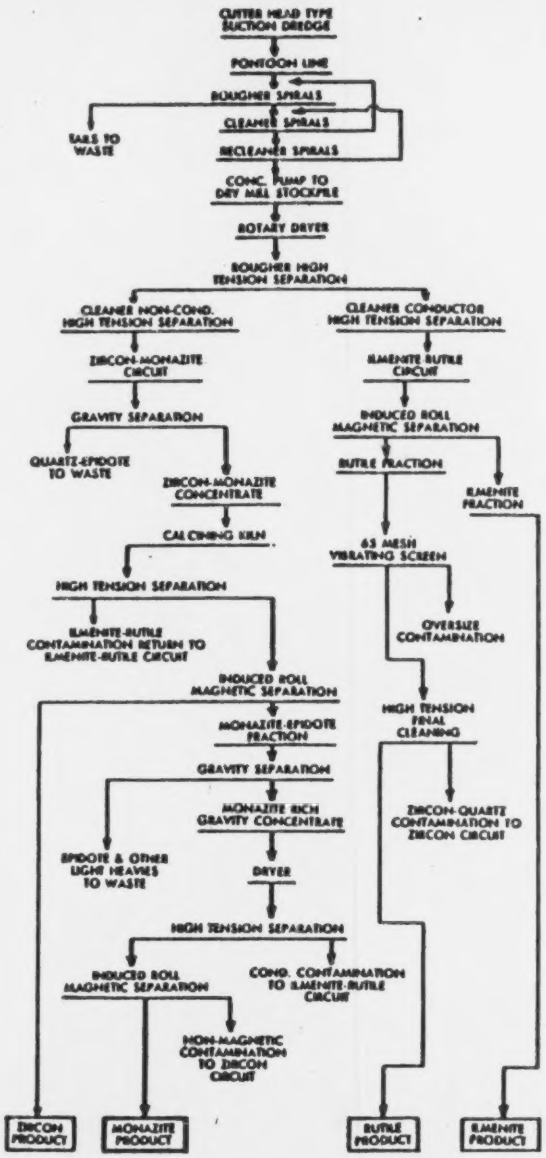

CUTTER HEAD TYPE SUCTION DREDGE
PONTOON LINE
ROUGHER SPIRALS
TAILS TO WASTE
CLEANER SPIRALS
RECLEANER SPIRALS
CONC. PUMP TO DRY MILL STOCKPILE
ROTARY DRYER
ROUGHER HIGH TENSION SEPARATION
CLEANER NON-COND. HIGH TENSION SEPARATION
CLEANER CONDUCTOR HIGH TENSION SEPARATION
ZIRCON-MONAZITE CIRCUIT
ILMENITE-RUTILE CIRCUIT
GRAVITY SEPARATION
INDUCED ROLL MAGNETIC SEPARATION
QUARTZ-EPIDOTE TO WASTE
ZIRCON-MONAZITE CONCENTRATE
RUTILE FRACTION
ILMENITE FRACTION
CALCINING KILN
65 MESH VIBRATING SCREEN
HIGH TENSION SEPARATION
OVERSIZE CONTAMINATION
ILMENITE-RUTILE CONTAMINATION RETURN TO ILMENITE-RUTILE CIRCUIT
INDUCED ROLL MAGNETIC SEPARATION
HIGH TENSION FINAL CLEANING
MONAZITE-EPIDOTE FRACTION
GRAVITY SEPARATION
ZIRCON-QUARTZ CONTAMINATION TO ZIRCON CIRCUIT
MONAZITE RICH GRAVITY CONCENTRATE
EPIDOTE & OTHER LIGHT HEAVIES TO WASTE
DRYER
HIGH TENSION SEPARATION
INDUCED ROLL MAGNETIC SEPARATION
COND. CONTAMINATION TO ILMENITE-RUTILE CIRCUIT
NON-MAGNETIC CONTAMINATION TO ZIRCON CIRCUIT
ZIRCON PRODUCT
MONAZITE PRODUCT
RUTILE PRODUCT
ILMENITE PRODUCT

Fig. 3. Typical flowsheet for processing titanium beach sands (from Lynd and Lefond, 1975, p. 1197).

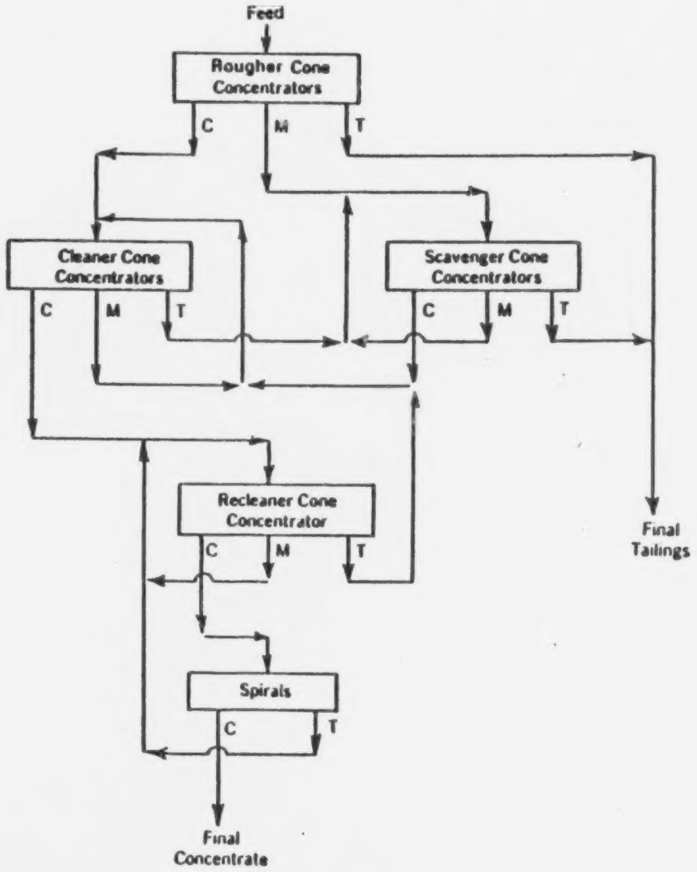

Feed
Rougher Cone Concentrators
C
M
T
Cleaner Cone Concentrators
C
M
T
Scavenger Cone Concentrators
C
M
T
Recleaner Cone Concentrator
C
M
T
Final Tailings
Spirals
C
T
Final Concentrate

Fig. 4 . Flowsheet of wet titanium mineral concentrator, Mineral Deposits, Ltd., NSW, Australia (from Aplan, 1985, p. 27-15).

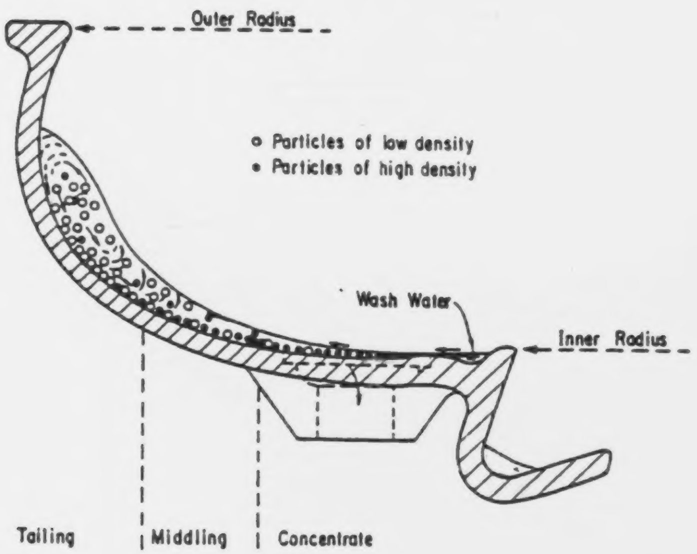

Outer Radius
o Particles of low density
• Particles of high density
Wash Water
Inner Radius
Tailing
Middling
Concentrate

Fig. 5a . Cross section of spiral trough (from Michell, 1985, p. 4-48).

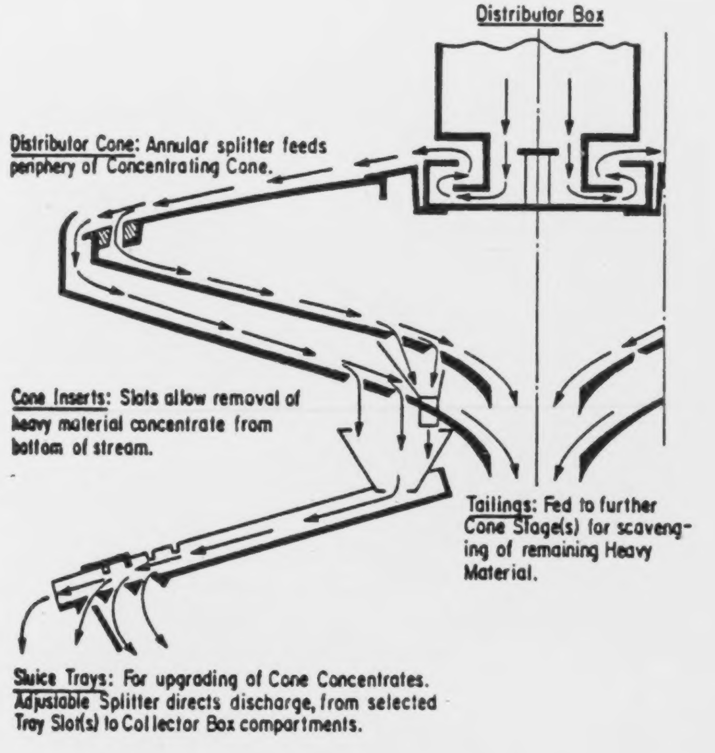
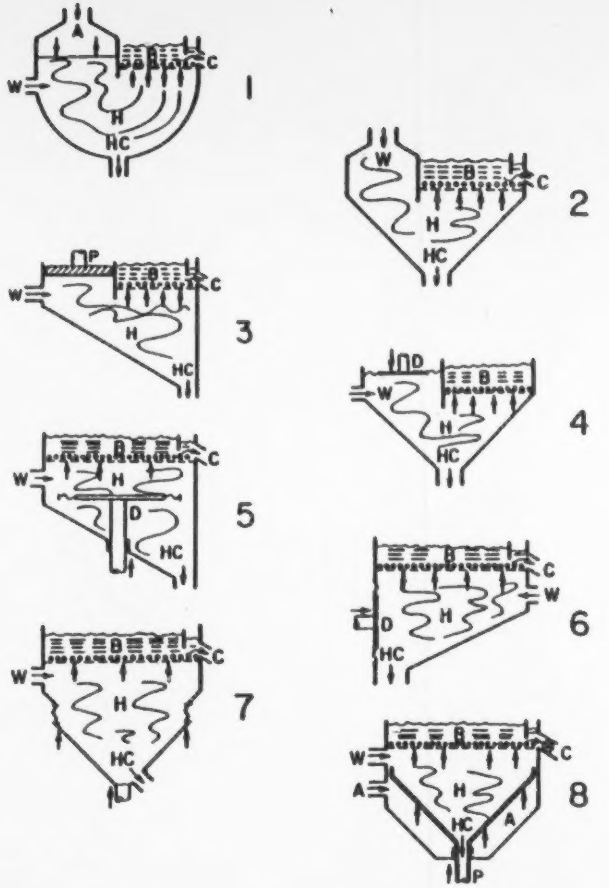

Distributor Box
Distributor Cone: Annular splitter feeds periphery of Concentrating Cone.
Cone Inserts: Slots allow removal of heavy material concentrate from bottom of stream.
Tailings: Fed to further Cone Stage(s) for scavenging of remaining Heavy Material.
Sluice Trays: For upgrading of Cone Concentrates. Adjustable Splitter directs discharge, from selected Tray Slot(s) to Collector Box compartments.

Fig. 5b . Cross section of the Reichert cone concentrator (from Michell, 1985, p. 4-47).


A
W
B
C
H
HC
1
W
B
C
H
HC
2
P
W
B
C
H
HC
3
D
W
B
H
HC
4
B
C
W
H
D
HC
5
B
C
W
D
H
HC
6
B
C
W
H
HC
7
B
C
W
A
H
HC
A
P
8

W=water inlet, H=hutch, B=jig bed, C=concentrate, D=diaphragm, P=piston. Type 6 includes Cleaveland IHC jig .

Fig. 5c . Fixed-bed jig types (from Pickett and Wiley, 1985, p. 4-29)

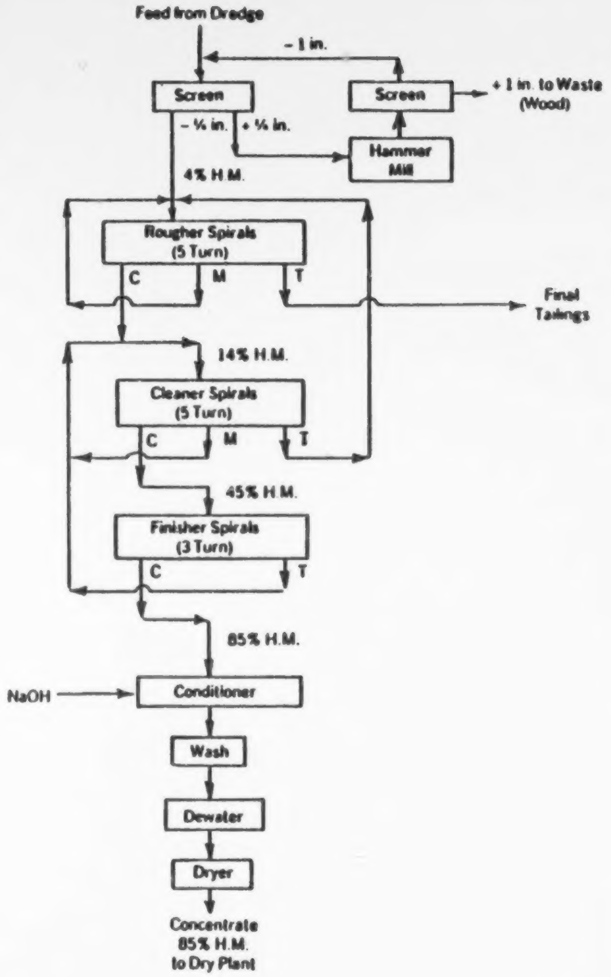

Feed from Dredge
- 1 in.
Screen
Screen
+ 1 in. to Waste (Wood)
- ¼ in.
+ ¼ in.
Hammer Mill
4% H.M.
Rougher Spirals (5 Turn)
C
M
T
Final Tailings
14% H.M.
Cleaner Spirals (5 Turn)
C
M
T
45% H.M.
Finisher Spirals (3 Turn)
C
T
85% H.M.
NaOH
Conditioner
Wash
Dewater
Dryer
Concentrate 85% H.M. to Dry Plant

Fig. 6 . Flowsheet of ilmenite wet mill, DuPont Trail Ridge, Fla. (from Aplan, 1985, p. 27-16).

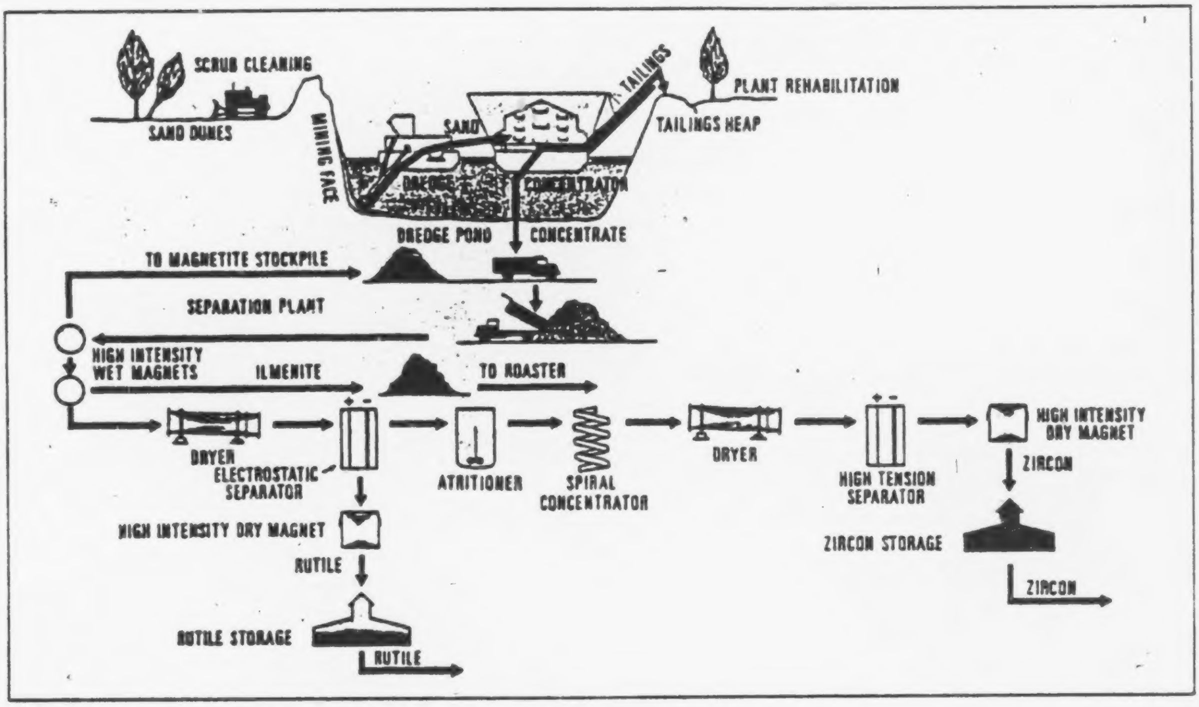

SCRUB CLEANING
SAND DUNES
MINING FACE
SAND
DREDGE
CONCENTRATOR
TAILINGS
PLANT REHABILITATION
TAILINGS HEAP
DREDGE POND
CONCENTRATE
TO MAGNETITE STOCKPILE
SEPARATION PLANT
HIGH INTENSITY
WET MAGNETS
ILMENITE
TO ROASTER
DRYER
ELECTROSTATIC
SEPARATOR
ATRITIONER
SPIRAL
CONCENTRATOR
DRYER
HIGH TENSION
SEPARATOR
HIGH INTENSITY
DRY MAGNET
ZIRCON
ZIRCON STORAGE
ZIRCON
HIGH INTENSITY DRY MAGNET
RUTILE
RUTILE STORAGE
RUTILE

FIGURE 7. Richards Bay Minerals Project — process flow diagram — mine.

Annual Volume, 1978 (Lee and Poggi)

Appendix

I. Methodology for Updating Costs.

Definitions, units and abbreviations:

M&S = Marshal and Swift, All-Industry Equipment Cost Index
$M = thousand U.S. dollars
$MM = million U.S. dollars
OC = operating costs
OH = overhead
tonne = metric ton (2204.6 lb)
st = short ton (2000 lb)
kW = kilowatt
kWh or kW·hr = kilowatt-hour

Example: wet mill costs for 1200 tonnes/hr of H.M. sands feed (mining rate 9.5 x 10^6 st/yr); 1984 data from IC 9061.

Capital cost, from Fig. 12 of IC 9061, is $13.6MM in 1984 $ (M&S index = 770).

Operating cost, from Fig. 14, is $3.1MM/yr in 1984 $.

From information given in the text of IC 9061, breakdown of operating costs is:

Labor - 54% of OC with rate of $23M per man-yr, including OH.
Power - 15% of OC with rate of 2.7-3.6¢/kWh; assume 3.15¢/kWh.
Maintenance materials - 31% of OC; consider this a fixed charge.

From the foregoing breakdown,

Labor = $1674M/yr (1984 $), or 73 personnel.
Power = $465M/yr " , or 2050 kW for 7200 hr per yr.
Matls = $961M/yr " , or 7.1% of fixed capital.

Updated capital and annual operating costs are computed as follows:

Current (late 1986) M&S index = 795; updated capital cost = 795/770 x $13.6MM = $14.0MM.

Labor - current U.S. @ $28M/man-yr x 73 staff	= $2,044M/yr
Power - current U.S. @ 6¢/kWh x 14,760 kWh/yr	= 886
Matls @ 7.1% of $14.0MM	= 994
	$3,924M/yr

Updated operating cost = $3.9MM/yr.

The same general methodology was applied to dry mill and general costs of the operation. A more rigorous cost updating procedure than that illustrated is not warranted at the present level of estimation or probable accuracy of data and assumptions.

2. Offshore Phosphorite Deposits: Processing and Related Considerations

Consulting and R & D in Extractive Metallurgy, Energy and Environmental Technologies

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OFFSHORE PHOSPHORITE DEPOSITS:
PROCESSING AND RELATED CONSIDERATIONS

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OFFSHORE PHOSPHORITE DEPOSITS

A. Background and Perspectives

1. Initial perspectives

The majority of U.S. economic deposits of phosphate are of marine sedimentary origin, as opposed to occurring in igneous rock. In spite of this broad commonality of origin, a variety of mining methods are in use depending, in large part, on the nature and thickness of the overburden and on the geological setting. Giant, electrically-powered walking draglines are a prominent feature of phosphate operations in the Central Florida district but are not appropriate for the majority of phosphate operations elsewhere. In the Western U.S. phosphate district, for example, one of the phosphate mines is underground. For the most part, mining methods used for land deposits of phosphorite are not suitable for offshore mining, although dredging is employed in conjunction with phosphate mining operations at certain locations.

A second point is that prices for "phosphate rock", as the beneficiated but not chemically converted phosphorite mine product is commonly called, are relatively low -- comparable to those of low and intermediate rank coals, for example. Current domestic prices for phosphate rock, f.o.b. mine, range from about $22 to $35 per metric ton (Stowasser, 1986), depending upon grade, or under 1 1/2 ¢ per lb.

Commercial phosphate rock has generally been separated from associated clays and silt-sized materials by a sequence of washing operations and from sand and other sand-size materials by flotation, usually followed by drying. Drying is always indicated when the phosphate rock is to be shipped to distant markets. Sometimes, calcination of the product is required to remove organic impurities preparatory to further processing or to increase the grade of the phosphate rock. Without adding in the direct costs and considerable problems associated with the treatment and disposal of typically voluminous clay wastes of the industry, it is obvious that there is little margin for recovering out-of the-ordinary capital and operating costs in a normal competitive market.

The clear implication for potentially profitable exploitation of offshore phosphorite is that the deposit be:

- world-scale in terms of minable tonnage and ore specifications.
- not overlain by a substantial layer of nonphosphatic sediments, i.e., the sediment blanket should optimally be practically non-existent, or not thicker than a small fraction of the depth of the mining cut.
- laterally continuous over large areas and of adequate and substantially uniform vertical extent, commensurate with efficient mining by dredge.
- located in relatively shallow water subject to sea, weather and current conditions that would not impede mining and offshore primary concentration during more than a small number of days out of the year.
- close to a suitable port or, at least, close to a shore where port facilities could be constructed at reasonable cost and also be within easy transportation distance to the onshore processing plant.

Qualitatively, most of the above requirements are obvious. Those responsible for the Mining study can provide specific limits on quantities such as mining depth, distance to port, etc. It is emphasized that provision of adequate port storage and handling facilities will generally be a nonnegligible item in the overall economics. Furthermore, the cost of providing storage and surge capacity at the port and at the onshore plant will be increased if offshore mining and primary concentration are impeded more than very infrequently by conditions at the mine site. Competing land-based operations run substantially continuously (so long as business conditions so warrant), with only a few days out of the year given over to scheduled holidays and maintenance.

Finally, the incentives, by most significant market and economic measures, have diminished since the 60's when commercial interest in offshore resources was high and active, and the 70's when phosphate rock prices rose to unsustainable heights. In particular, the lessened foreign demand for U.S. phosphate rock exports (except for the Canadian market) and high capital requirements make any major new phosphate mining venture in the U.S. economically unattractive. These changes are unlikely to be reversed in the near future.

Considerations such as these foregoing probably account for the omission of offshore phosphate deposits from the five-nation cooperative inventory of major world resources of phosphate (Krauss et al., 1984); in that study, offshore deposits were classed with "the vast identified subeconomic resources". Similarly, the thorough Phosphate Rock Availability appraisals (Fantel et al., 1983, 1984) merely note that "New deposits will likely be discovered in the U.S., particularly offshore deposits along the eastern seaboard."

2. Onshore/offshore comparisons

Competition and conflicts relating to land and aquifer use are cited as incentives for moving the operations offshore, where maintenance of large impoundments for extremely slowly settling phosphatic clay "slimes" would not come into question. Land reclamation costs in the Central Florida district now run about $4000 per acre (T.P. Oxford, pers. comm.). On the other hand, it is not assured that disturbances attending dredging and on-site disposal of the waste fines will be acceptable. This is discussed in the chapter by OTA staff on environmental issues. It has been reported that exploration of the heavy mineral sands lease area off Georgia was held up for 6 months because the area is used as calving ground by right whales, an endangered species.

Fresh water consumption in phosphate rock production would, of course, be greatly diminished by replacing present hydraulic disaggregation, slurrying and washing of the ore by offshore dredging and primary concentration employing seawater. It is even conceivable that use of sea water for these purposes would result in less dispersion of the clay and silt, but this would need to be demonstrated.

Interference of sea water, contained in the pebble-size phosphate rock fraction, if any, and in the sand-size concentrate, with subsequent use or processing would be anticipated. Without reference to specific sites and circumstances, it would probably be advantageous to remove sea water from the phosphate rock and concentrate by fresh water washing at the port storage. In consideration of the prior removal of the fines and of the inertness of the sand and phosphate rock, the effluent from the fresh water wash should be nothing more environmentally hazardous than diluted sea water. There is now a precedent for concentrating phosphate ore on a large scale with use of sea water, as will be described in Section C.

An obvious critical factor in the assessment of relative attractiveness of onshore and offshore phosphorite deposits is the content of residual impurities in the washed and concentrated phosphate rock. Specifications for commercial phosphate rock include maximums for $CaO:P_2O_5$, MgO, Fe_2O_3, and Al_2O_3. Presently, insufficient resource information is available for the main known offshore occurrences, namely, on the continental shelf off Southern California and on the southern half of the Atlantic Continental Shelf, to make explicit comparison here. A number of general and detailed chemical analyses have been made of individual offshore phosphorite samples, and some of this information on composition has been published. For example, Manheim et al. (1980) tabulate partial analyses for some phosphorites of the Blake Plateau, off the Coast of South Carolina (Tables 1, 3).

Inderbitzen et al. (1970) give chemical analyses for 20 phosphorite samples (defined by them as those samples which analyzed over 21% P_2O_5) out of 51 sea-floor samples collected off Southern California. Their average analysis of the phosphorite sample group has been cited by others, including Chambers Consultants and Planners (1979), and is shown in Table 2. The term LOI means "loss on ignition", i.e., loss in weight on heating to red heat in air, and measures the combined presence of materials that are decomposed, vaporized or combusted by such treatment; for phosphorites, LOI measures mainly organic materials, CO_2 from carbonates, and water. The letters BPL denote the quaint common name "bone phosphate of lime" or calcium orthophosphate, $Ca_3(PO_4)_2$ (or $3CaO.P_2O_5$), the stoichiometric and unsubstituted formula for the generic mineral, apatite. The equivalence is: pure $Ca_3(PO_4)_2$ = 100% BPL = 45.76% P_2O_5 = 20.0% P.

Additional chemical analyses of offshore phosphorite samples of the Southern California Borderland were compiled and evaluated by Hess (1979). Among other analyses, Hess displayed data by Pasho showing an inverse linear relationship between %P_2O_5 (ca. 22-32%) and % insoluble (ca. 21% to >1%) and a roughly linear increase of %F (ca. 2.5-3.5%) over the same range of %P_2O_5, for six areas of the Borderland. The latter observation is consistent with the generalization that sedimentary as well as igneous phosphorites contain fluorine in the rough proportion of 1F to 10 P_2O_5 (Emigh, 1975). The major phosphate minerals are, respectively, francolite, (carbonate fluorapatite containing up to about 2% structural CO_2) and fluorapatite. Detailed chemical analyses, but excluding trace elements for the most part, are given in Table 4 from Emigh (1983) for concentrates of marine sedimentary phosphorites from a variety of sources, including a sea-floor nodule from Forty-Mile Bank (Location Code #22) and uncalcined, beneficiated beach sand from Baja California Sur (Location Code #19). That Mexican beach deposit will be considered further, below.

As illustration of the divergence of actual carbonate fluorapatites from the composition of stoichiometric fluorapatite, namely $Ca_{10}(PO_4)_6F_2$, consider the structural formula derived by Manheim and Gulbrandsen for an average Florida carbonate fluorapatite in 30 concentrate samples, viz., $Ca_{9.7}Mg_{.1}Na_{.2}(PO_4)_{5.1}(CO_3)_{.7}(CO_3F)_{.2}F_2$. Table 5 gives their chemical analyses of the apatite constituents of phosphorite concentrates from five areas, including the continental shelves of South-West Africa and Peru. In Table 5, WUSA refers to the Western phosphate district. The corresponding analyses in terms of BPL range from 45% BPL for the offshore Peru samples to 74% BPL for samples from Morrocan deposits. For comparison, BPL ranges and listed prices of commercial phosphates are shown in Table 6 from Industrial Minerals.

A newly published, comprehensive survey of subsea mineral resources (McKelvey, 1986) includes informative consideration of phosphorites. In it, the author again (cf. McKelvey, 1968) notes that offshore phosphorites are unlikely to be as rich in P_2O_5 as are weathered and reworked deposits of marine origin found on land. The latter may contain as much as 31-35% P_2O_5. Although individual subsea samples occasionally analyze 30% P_2O_5 or higher (Table 3 of McKelvey, 1986), average assays seldom exceed 29% P_2O_5 (63% BPL). Information presented by Mullins and Rasch (1985) on central California continental margin phosphorites is consistent with this generalization. For their samples, P_2O_5 content ranged from 11.5% to 31.0% and averaged 24.4%. They conclude that the patchy distribution and relatively great water depth (200 to 2000 meters) of the central California margin phosphorites make them economically less attractive than the deposits offshore southern California, discussed in the next section.

A recently discovered and relatively intensively explored and analyzed continental shelf phosphate province in Onslow Bay, North Carolina (Riggs et al., 1985) gives evidence of containing deposits of higher than typical grade, in major respects. For two large districts, single-flotation concentrates averaged 29-31% P_2O_5, after adjustment of analyses to zero percent insolubles. Further exploration and deposit delineation are clearly warranted for this region. At an appropriate stage, testing of representative (three dimensions) samples of meaningful size, for amenability to mineral processing and fertilizer manufacturing technologies, will provide the basis for a partial process engineering and economic analysis. The word 'partial' is employed, because great uncertainty will probably be still attached to the technology and costs of all required offshore operations, including dredging, preliminary beneficiation, waste disposal and mine site restoration.

The reduced demand of recent years for domestic phosphate and projections of underutilization of capacity for some years into the future could be seen as providing opportunity for further exploration and delineation of phosphorites of the EEZ. Additional notably promising deposits may well be found to add to the potential phosphate resource. The concern expressed by many in recent years, including by Riggs et al.(1985), that increasing demand for phosphate, coupled with conflicting land-use pressures, will drive up the cost of producing land-based reserves, will doubtless again acquire immediacy. That time will come sooner if shut-down plant capacity becomes uneconomic to reopen and shut-down mines are irrevocably diverted to other land uses.

3. Development incentives

Reconnaissance, sampling and analysis as illustrated above, and associated amenability testing are essential, in a normal business climate, to stimulate interest in possible commercial development. However, even the most favorable indications to this stage will not generally have influence beyond a decision to look further into a given occurrence as a potentially commercial resource. In addition to the factors usually mentioned in this connection, dredging of sufficient representative material for pilot plant testing, as was done for manganese nodules, will generally be a necessary preliminary to the business decision to invest substantial sums in delineating the deposit adequately for a subsequent decision with respect to mining. Delineation requires determination of the vertical as well as lateral extent and continuity of the deposit by systematic and relatively close-spaced drilling or coring, preferably supplemented by selected test dredging to the underlying stratum.

All of this is rather obvious and is well understood by those involved in EEZ exploration and resource evaluation. The point is that none of the placer minerals, excluding barren sand and gravel, or phosphorite deposits under consideration in the OTA study have been delineated to the degree that an informed decision can presently be made with respect to a recommendation for commercial development. It becomes understandable why, based on the preliminary resource information available, some suggest that certain resources are economically minable, while others argue that the economic incentive does not exist. Bowen (1972) and Riggs (1986), for example, take a generally optimistic stance with respect to offshore phosphorite, whereas Sorensen and Mead (1969) and Manderson (1972), among others, concluded that the economics were definitively unfavorable.

In spite of unfavorable indications from prior commercial activities, enactment of the Outer Continental Shelf Hard Minerals Leasing legislation led to a series of Interior Department and contracted studies on OCS mineral resources, including three reports on offshore phosphorites. Of these, the preliminary resource assessment by Hess (1979) of phosphorites of the southern California borderland draws the inference that his informative and professional study "has indicated sufficient resource potential to justify further detailed sampling/delineation of most promising deposits and to generally set forth a rationale for mining of marine phosphates on a commercial scale." This position, as we interpret it, of exploring and evaluating the resource further, against the backdrop of trends in the phosphate industry and markets, appears to be in consonance with the views of many workers in the field of marine mineral resources. See, for example, Broadus (1985) and the report and recommendations of the NATO (1986) Working Group on Marine Phosphorites.

4. Past Commercial activities

A major impetus for commercial consideration of phosphorite deposits off southern California was the presumed advantage in transportation costs to West Coast markets, as compared with shipping by rail from the Western phosphate district or by ship from the Southeastern phosphate district. Chambers Consultants and Planners emphasize this point in their 1979 study.

As reported by Inderbitzen et al. (1970), Lockheed Aircraft Corp. and International Minerals and Chemical Corp. jointly conducted an intensive reconnaissance, sampling and laboratory study of phosphorite deposits offshore southern

California during 1963 and 1964. The main factor in their apparent decision not to proceed further toward commercialization was said to be the finding of lower average P_2O_5 content and higher ratio of detrimental metallic impurities to P_2O_5, even for hand-picked nodules and pellets (Table 2), as compared with phosphorite concentrates from domestic land-based operations.

Preceding and possibly overlapping the commercial activities just described, Collier Carbon and Chemical Company are said to have actively explored and dredged in a lease area on the Forty Mile Bank off San Diego. The results are thought by informed sources (see, for example, Sorensen and Mead, 1969) to have been disappointing both with respect to the quality and quantity of phosphorite nodules dredged and to the difficulties encountered in dredging in unsheltered waters, and the leases were relinquished.

In the late 1960s, the most promising North American offshore phosphorite deposits were considered to be the sand deposits of Santo Domingo Bay, Baja California (Overall, 1968). A combination of Mexican and U.S. interests, including the Global Marine Co., carried out a program of exploration and shallow-core sampling in the primary area (water depth 50-150 m). According to Overall, P_2O_5 grade in place ranged from 5 to 12%, with the bulk of the nonphosphate material being in the clay-size fraction. Initial laboratory tests indicated that a simple sizing operation could beneficiate the ore to 16-18% P_2O_5, the major diluent materials being detrital silica sand and sea shells. It was thought that flotation and calcining could produce a concentrate grade of 31% P_2O_5. Apparently, exploitation of this offshore resource was postponed indefinitely, in favor of an extensive, but lower grade on-land phosphorite sands deposit in the Santo Domingo area. The latter development will be considered in Section C.

B. Qualitative Economic Considerations

1. Broad cost comparison

The remaining two reports in the OCS phosphorites series describe and cost offshore mining and processing scenarios, one for each coast. In these reports, Chambers Consultants and Planners (1979) and Zellars-Williams, Inc. (1979), specific mine site and assumed or inferred resource characteristics form the basis for the estimation of economic feasibility. However, without developing a specific plan for mining and processing, one can make some broad cost comparisons, as follows.

It is assumed that dredging, any necessary screening and disaggregation, and washing, i.e., mining and primary concentration, will be performed offshore; clay and silt-size material will be discarded directly to the mined-out area, and the pebble-size and unseparated sand-size phosphorite brought ashore for further processing. If one enumerates all of the steps, beginning with exploration and leading to final processing on shore, and makes a very qualitative step-wise comparison of associated major costs, the conclusion becomes evident that the economics to that point favor an equivalent resource on land. This view is reinforced by the likelihood that investors would expect a shorter payback and higher rate of return, on account of uncertainties associated with mining offshore -- perhaps double the 15% DCFROR usually employed in mineral availability studies. Sorensen and Mead (1969) made this point with respect to exploitation of offshore phosphorites.

2. Cost offset via higher ore values

Accepting the results of the foregoing crude cost comparison, one concludes that the offshore resource would need to possess some strongly advantageous feature(s). Possibilities that come to mind include higher grade, meaning both a larger percentage of P_2O_5 (or equivalent BPL) content and a correspondingly lower content of inert (diluent) and specifically deleterious impurities. Another advantageous feature, not characteristic of phosphate ores on land, would be the presence, in separable amounts, of high-value accessory minerals. By "high value" are implied minerals such as rutile, zircon and monazite, and not minerals such as ilmenite and chromite, whose prices as concentrates are comparable to those of phosphate rock.

A feeling for the percentage of higher-value accessory minerals that would be required to make a significant improvement in the economics of mining offshore phosphorites is obtained by considering present prices of concentrates. At $320 and $26 per metric ton of rutile and 66-70% BPL phosphate rock, respectively, rutile would need to comprise about 7 1/2% by weight of total recovered phosphate rock plus rutile in order to yield co-product revenue equal to that of the phosphate rock. Correspondingly greater and lesser percentages would pertain, respectively, to co-product zircon at $140 and monazite at $530. Much smaller percentages of higher-value accessory minerals than those suggested would probably not repay the incremental cost of their recovery. It will be noted, in this connection, that entirely different beneficiation processes are employed for phosphate and heavy mineral ores, so that the postulated co-product recovery would entail major modifications to the phosphate recovery operation.

The economic picture with respect to offshore phosphorites could be analogous to that for ilmenite mineral sands, the mining of which, for the most part, is not profitable unless one or more marketable co-products of higher unit value are also recovered.

Neither higher phosphorite grade nor recoverable co-products are evident in available analyses of Pacific or Atlantic continental shelf phosphorites. On the question of detrimental impurity contents, as relating to market grade, fluoride is evidently comparable for the offshore deposits; chloride would certainly be greater, unless the product is comminuted and thoroughly washed; and carbonates and organic materials are likely to be high or, at least, on the high side compared with commercial phosphate rock. Appreciable iron and manganese impurities as, for example, in the form of ferromanganese oxide staining and impregnating phosphorite nodules and pavements in areas of the Blake Plateau, would also be objectionable in a phosphate pebble or concentrate product.

C. A Phosphorite Placer Development

There appear to be only two phosphorite placer deposits in production or preproduction. These are the Langebaan deposit, discovered in 1897, at Salardina Bay in South Africa and the Santo Domingo development in Baja California. Mining at those locations is conducted by dredging. According to abbreviated production history given by Krauss et al. (1984), Langebaan has become only marginally economic: about 29,000 tonnes (t) of 29.5% P_2O_5 concentrate (64.5% BPL) produced in 1981 from average 8.6% P_2O_5 ore as compared with an average annual production of about 0.7 x 10^6 t for 1943-82. By contrast, the large igneous, or magmatic-origin, ore body at Phalaborwa averages 7% P_2O_5 and is mined at a rate to produce about 3 x 10^6 t of 36.5% phosphate concentrate per annum (Fantel et al., 1984).

Recent information is available on the Santo Domingo pre-mining development (Gruber et al., 1986) which will, apparently, become the closest real-life analog of a major offshore phosphorite operation. It is, therefore, pertinent to consider its salient engineering and economic features.

Testing and cost estimates by Zellars-Williams Co. indicate that production from the Santo Domingo ore will be economic in a normal phosphate rock market (Gruber et al., op. cit.). Thus, there is a strong incentive for Mexico, a net importer of phosphate rock, to complete the development. The current degree of import dependence can be gauged by noting that the anticipated 1.3 x 10^6 t annual production of concentrate from Santo Domingo, when combined with the current production of about 0.7 x 10^6 t from the San Juan de la Costa open-pit operation (also located in Baja California Sur, but not a beach deposit), will provide about 80% of the country's phosphate rock consumption.

Economically exploitable, in this case, can be taken to mean that the cost of the product as delivered to the fertilizer complex in Mexico is not expected to exceed the price of comparable U.S. phosphate rock shipped from a Gulf port and including the considerable shipping charge to the Mexican plant. For example, the shipping charge between the ports of Tampa and

Veracruz was $11.20 per tonne in 1981 (Fantel et al., 1984). Trade balance, foreign exchange and national self-sufficiency are all factors in the decision to proceed. It may be remarked that similar considerations in the U.S. have not generally been major factors in business decisions affecting exploitation of nonfuel mineral resources.

The revised flow sheet calls for recovery of 70% of the P_2O_5 in the flotation feed to produce 1.3×10^6 t of 29% P_2O_5 (63% BPL) at an ore-to-concentrate ratio of 14.1 to 1. This is an unusually low grade ore by world mining standards. The estimated mining rate is $(14.1)(1.3) \times 10^6$ t = 18×10^6 t, or about 20 million short tons per year. The overall P_2O_5 recovery is 64% $(1.3 \times 10^6 \times 29) \div (18.3 \times 10^6 \times 3.2) = 0.64$; the recovery of P_2O_5 in the primary concentrator feeding the flotation plant (recovery 70%) is estimated to be 0.64/0.70 = 0.91, or 91%.

The overall estimated P_2O_5 recovery, from mined ore through final concentrate, is comparable to that for conventional sedimentary phosphate operations, where about a third of the phosphate values is typically lost to the waste clay slimes and as apatite and nonapatite phosphate minerals in the flotation tailings. This result is brought about by compensation between a higher than usual recovery in initial size separation and a lower than usual recovery in flotation. The beach deposit consists largely of unconsolidated mineralized sand and clay strata, the phosphorite occurring from the surface (i.e., substantially no overburden) to a depth of 21-33 meters, bottoming out at about 15 meters below mean sea level (Gruber et al., op. cit.). The ore zone is 20 m thick on average.

The general mining and beneficiation scheme at Santo Domingo includes dredging the ore and preparing a rougher concentrate on a trailing floating concentrator (Fig. 1a) and pumping the rougher concentrate to the land plant (Fig. 1b) for conditioning and sequential flotation to remove gangue materials (principally sand, silicates, carbonates). The product is to be shipped by barge to Porto Belcher, a deep water port ca. 70 km away (Williams et al., 1985).

Several features of this deposit and operation are instructive for future offshore mining of phosphorites in largely unconsolidated sediments. The roughly 2% value-mineral content (in this case, P_2O_5) in fine to medium grain sand with low silt and clay content is reminiscent of minable H.M. sand deposits. Two cutterhead dredges are employed, for an average mining rate per dredge of 10 million (short) tons of ore per year, rather typical of heavy minerals sands dredging. At an assumed stream factor of 90% at Santo Domingo, i.e., about 330 days of operation per year, actual dredging rates would be about 30,000 tons per day or about 1250 tons per hour per dredge.

In the mine development phase, hard strata were unexpectedly encountered, which necessitated further core drilling for their delineation. The cemented layers are higher in P_2O_5 than are the unconsolidated sands so that modifications to the beneficiation as well as mining facilities, which had been designed for unconsolidated sediments, were required. All of this has been time-consuming and expensive and adds dimension to our earlier emphasis on adequate drilling and test dredging in the offshore prior to a mining decision.

A third feature of significance is that, whereas conventional phosphate beneficiation operations are copious consumers of underground water, there is apparently no fresh water naturally available at the Pacific Coast location of the extensive Santo Domingo phosphorite province in central Baja California Sur. Consequently, ocean water is used to maintain the dredge pond level and for all processing steps except for the fresh water wash of the final concentrate (Fig. 1b). According to the Zellars-Williams authors, the fresh water is obtained by reverse osmosis desalination of sea water, and consumption is minimized by employing three stages of counter-current washing. It is essential to remove residual sea salts in the final concentrate in order to avoid severe corrosion problems further downstream at the wet-process phosphoric acid plant of the fertilizer complex. In this case, product specification is 350 ppm Cl, maximum.

The unavoidable use of seawater reduced the effectiveness of the flotation separation following size separation and required further modification of the conventional flow sheet. This complication could be avoided for an offshore operation with a land plant at a location having adequate fresh water for processing. This ~~has~~ been assumed, at least implicitly, when exploitation of shelf phosphorites has been considered. For the offshore case, the beneficiation of the phosphorite on the dredge (or floating concentrator) would be carried only through size separation (Fig. 1a), with the primary concentrate being transported to the land plant. This avoids having to attempt a flotation separation at sea and, thereby, avoids the immediate introduction of conditioning and flotation reagents into the marine environment. The oversize and fines (undersize) returned to the bottom will be essentially in their original states, except for having been segregated by size.

If, then, adequate aquifer capacity is available at the land plant, and if the composition of the primary concentrate falls within normal limits, a conventional flotation sequence (Fig. 2) can be used, as proposed in certain scenarios for offshore phosphorites.

The consideration of heavy mineral (H.M.) content in the Santo Domingo ore is instructive for evaluation of offshore deposits. Mineral analyses of -14,+150 mesh ("sands") and -150,+400 mesh ("slimes") size fractions of the ore are given in Table 7. Note that the size cut-off for desliming in the adopted flow diagram (Fig. 1a) is 74 microns (200 mesh).

The H.M. suite includes magnetite, ilmenite, garnet, zircon and others. Far from increasing potential revenues, the H.M. content at the level shown requires additional processing steps for its removal. Unremoved, the heavy minerals constitute "insolubles", or "insols", in the final concentrate and would unacceptably dilute the phosphate values.

In the prior Santo Domingo flowsheet, carbonate and H.M.'s were removed by gravity separation techniques. This resulted in low phosphate recovery and selectivity (Williams et al., 1985) Appreciable percentages of high value heavy minerals would presumably lead to serious consideration of their separation and recovery, as discussed above. In general, however, the success of phosphorite mining hinges on the grade of recoverable phosphate and not on potential by-products.

If the extensive (ca. 1600 km^2) Santo Domingo phosphate province were, in concept, moved offshore onto the shelf, the presently anticipated economic return would, in our judgement, become unfavorable owing to added direct and indirect costs without compensating revenues. The added cost factors have been mentioned in the general context of offshore mining but, briefly, include investment in seagoing dredge and primary concentrator; incremental equipment, crew and facilities expenses due to the nature of shipboard operation, including a probably lower stream factor; and expenses associated with the port facility.

Two prior engineering economic studies of offshore phosphorites, which assumed deposit characteristics and estimated economic feasibility therefrom, will be analyzed briefly in the following section. The report of a third, earlier study has been reviewed and our analysis thereof added to the text of Section D.

D. Prior Engineering/Economic Studies

Two reports, each prepared for the U.S. Geological Survey under the Outer Continental Shelf Hard Minerals Leasing program and dated August 1979, summarize economic feasibility assessments of hypothetical offshore phosphorite deposits. As outlined in Table 8, the Chambers Consultants and Planners (CC&P) study dealt with a generic deposit at an unspecified location offshore southern California. The Zellars-Williams (Z-W) study considered a designated 30 square mile block located about 12 miles offshore Tybee Island, Georgia (close to the border with South Carolina), at a mean water depth of 42 ft. Salient features of the CC&P and Z-W cases are compared in Table 8 with corresponding actual information on the Baja California phosphorite beach placer development just described.

As a major part of the present assignment, the two engineering/economic reports have been scrutinized. The Z-W report reflects the company's competence and experience in substantially all phases of the phosphate industry, and it was thought worthwhile to update their cost figures to current dollars and to comment on major assumptions and conclusions. The CC&P report, on the other hand, does not hold up well under scrutiny, as inspection of the limited information given in Table 8 will suggest. It is considered, therefore, that prior evaluations of offshore southern California phosphorite deposits by commercial interests, as described above, are more credible in their treatment and conclusions.

The Z-W authors made clear that their mining and processing plan was based on a deposit whose characteristics were determined by geological inference, supplemented by a few samples and one Army Corps of Engineers drill core taken outside of the designated mine area. In that context, their cautious assessment that the Z-W study indicated "technical and economical feasibility to a degree sufficient to justify further evaluation and stimulate commercial interest", with considerable development required for accurate evaluation, is reasonable. However, their numbers are probably optimistic, particularly in view of conditions in the industry subsequent to the report.

In-house amenability tests with samples from the designated drill core supported the concept of a relatively simple mining plan and conventional mineral processing sequence. A simplified flowsheet for a typical phosphate rock operation in the Southeastern U.S. is shown in Figure 2, for comparison with the Z-W plan for offshore mining and primary concentration.

Among the favorable advantages assumed or inferred for the mine site and onshore plant location are:

a. only a few feet of sandy overburden and an adequately thick and supposedly continuous matrix (phosphate ore) zone -- see Table 8 data -- amenable to mining by cutterhead dredge.

b. a relatively short run to an adequate port facility at Savannah.

c. availability at the onshore plant site of an adequate supply of river water for process use, including washing of sea salts.

The Z-W mining/processing plan may be summarized as follows:

dredge mining	overburden removal and relocation; matrix mining, lifting to onboard primary concentrator
onboard mineral processing	mechanical disintegration of matrix, followed by size separation: screening (oversize rejected) and cycloning (undersize, mainly clays , flocculated and returned to bottom)
onshore processing	sand-size material subjected to further washing and sizing (rod mills); tailings and clays returned to mine site for placement over the flocculated clays; phosphate values concentrated by a conventional anionic,

cationic flotation sequence to 66% BPL; wet concentrates blended and calcined to 68% BPL.

Note, in the above comparison, that pebble-size phosphate rock was not thought to be a major constituent of the offshore deposit and would not be produced by the above mining plan. There is ample precedent for this in the southeastern U.S., in that for phosphate operations in North Carolina and in the South Florida district, for example, the major product is flotation concentrate, rather than pebble.

The Z-W component cost figures have been revised to reflect subsequent changes in plant, equipment, wages and other cost factors. The revised figures are summarized in Table 9 in current (1986) dollars. Before commenting in detail on the revised costs, some major questions and uncertainties are noted. Foremost, perhaps, is the adequacy and use of sea curtains to mitigate anticipated environmental effects of bottom dredging and of tailings and clay waste disposal. Related to this, and anticipated in the Z-W report, is the question of acceptability of at-sea disposal of the flocculated clays, especially from the standpoint of the presence of the chemical flocculants.

For these and other reasons related to a pioneering offshore venture, a DCF return much larger than the 16.5% return indicated in the Z-W study, for the case of a start in 1986 at a lower initial rate of 2.5 million tons of concentrate per year, would seem warranted. However, this asserted expectation of higher return on the part of investors has not been quantified. What is clear is that phosphate rock prices have not risen as anticipated in 1979, namely, from $18.00 per ton calcined rock, 68% BPL (f.o.b. Savannah, Georgia) in 1979 to $24.00 per ton in 1986, to $34.00 in 1996, all in constant 1980 dollars. By contrast, present price (Stowasser, 1986) for 68% calcined rock, Florida and North Carolina is about $21 per ton; in other words, real prices have declined appreciably. See, for example, Stowasser (1985) for a terse account of the hard-pressed state of the domestic phosphate industry. These circumstances translate into greatly reduced incentive to make the considerable investment required for developing hard data on potentially minable offshore deposits, preliminary to a decision on mine development.

Another Z-W study assumption, having a significant influence on estimated costs, is the 80% operating factor, or stream factor (80% = 292 days = 7008 hours). As applied to operation of the onshore plant, 80% may be conservative but, as applied to the dredge and washer, it may be optimistic. This would depend upon whether the dredge and onboard primary concentrator could function efficiently for ca. 7000 hours per year, given the sea and weather conditions at the mine site. The separation efficiencies of the hydrocyclone washers and the clay-flocculation clarifier may be reduced well below design and require motion compensation for improvement.

The largest component of total investment cost and of total operating cost in the Z-W evaluation is processing the primary concentrate, at the land plant, to 66% BPL phosphate concentrate (Table 9). The second largest cost component is calcining to 68% BPL. The technologies and, therefore, the costs of these established operations are well defined. The economics of calcining are particularly sensitive to fuel costs; even at the current rather depressed prices for fuels, energy requirements for the calcining step account for about 80% of operating cost for that step. Costs for the mineral processing sector are also unlikely to decrease below the figures presented here and could well increase by up to 20% or so, if a more complex flotation flowsheet were found necessary to achieve grade and recovery targets. In this connection, recall the revisions required to the Santo Domingo flowsheet and the earlier mention of apparent tendency toward higher values of some common detrimental impurities in offshore phosphorite deposits.

Transportation costs are site specific, and the approximately $2 per ton figure for transport between hypothetical mine site and closest major port is likely to be a minimum value, consistent with the short distance involved. The overall Z-W scenario also contains the implicit assumption that no major port development work would be required to support the operation. If such were not the case, project capital requirements might be increased by 5% or so, but operating costs would not be greatly affected.

Mining and primary concentration cost estimation included the assumptions that only 4 yd^3 of ore would need to be dredged per ton of final product (matrix "X" factor) and that the stream factor would be 80%. Lower separation efficiencies and recoveries in primary concentration, higher than anticipated maintenance requirements in offshore operation, and lower ore grade or recovery could increase the required dredge capacity, the size of the onboard washer plant, and the amount of surge capacity required. The economies of scale in the construction of the dredge, its platform and the onboard separation plant are such, however, that capacities could be increased by about a third, with an increase in capital requirements and operating costs of less than about 10% and 5%, respectively.

Overall, the scenario and associated costs developed in the Z-W study can be regarded as a "best-case" situation. In current dollars, the capital and operating costs to mine and wash the ore and to transport the primary concentrate to an onshore process plant are estimated to be $13.5 per annual ton and $4.8 per ton, respectively. Processing costs and working capital add an additional $39.0 per annual ton and $10.1 per ton. These are estimated cash costs and do not include capital recovery and profit, nor freight costs for moving the product from the process plant to market. Under current tax law, depreciation expense would amount to about $9.7 per ton of product.

From the above cost estimates, the break-even price of the calcined concentrate, f.o.b. plant, is seen to be almost $25 per ton. As noted earlier, this is in excess of recent posted prices for 66-68 BPL phosphate rock. Current prices probably come close to representing marginal costs of production from established mines, through plants that have already been depreciated.

Based on the present and projected near-term cost/price relationship, it is not possible to justify, on purely economic grounds, any major new phosphorite mining and processing venture in the southeastern U.S., whether the deposit is land-based or located offshore.

For offshore Georgia/south Carolina deposit evaluated by Zellars-Williams, the costs of mining, washing and transportation to port, including capital recovery, are over $14 per ton of product. A venture to produce phosphate from this source could not, then, be sold on the basis of availability of a low cost ore body. We arrive back at the conclusion that what is required by way of investment incentive is an improvement in the domestic phosphate rock market, at least to the level envisioned in the Zellars-Williams study of seven years ago.

The economic evaluation by Trondsen and Mead (1977) of California offshore phosphorite deposits was obtained too late for our review thereof to be included in the preliminary draft chapter. Their broadly informative study is more substantive than that reported by Chambers Consultants and Planners (1979). Both reports are concerned with production of the same general resource, namely, the major phosphorite deposits of the southern California borderland. See Supplement to Table 8, on Page F-6, for an outline of the Trondsen and Mead evaluation scenario. By comparison with the evaluation of an offshore Georgia and South Carolina deposit by Zellars-Williams (1979), the latter's report provides more detailed information in support of estimated costs than does the Trondsen and Mead report.

The economic evaluation in the 1977 report is based on the mining of phosphorite nodules by use of either conventional hydraulic suction dredging or an advanced type of jet-lift dredging system. It is not known by us whether or how far development of the latter has proceeded. The range of water depth for all eight identified deposits in the study area is reported as 4 to 700 meters, with average depths ranging from 75 to 400 meters. Assumed mining depth is 750 ft (230 m); assumed transport distance from mining station to the Long Beach harbor is 80 nautical miles.

Mining operations would be conducted on 200 days of the year. Actual dredging would take place during 20 hours of the day, or 4000 hours per year (a roughly 45% operating factor). In the base-case deposit characterization, the dredged ore consists of

2/3rds separable phosphorite nodules and 1/3rd "debris", mainly silt and sand, which is rejected overboard.

The mined phosphorite nodules would be transported to port in chartered barges and minimally beneficiated in an onshore plant by crushing, screening and calcining. The calcined product would be sold as low grade phosphate rock, with an added deep discount for residual impurities that are normally removed in beneficiation. Production rates of 1×10^6 and 4×10^6 st/yr (from 1.5×10^6 and 6.0×10^6 st/yr, respectively, of mined ore) were compared by Trondsen and Mead for the two dredging technologies. Updated costs of the most attractive option, namely, mining at the rate of 4×10^6 st/yr of phosphate rock with use of a jet-life dredge, are summarized in Table 10.

The updated Trondsen and Mead costs are tabulated in a format that allows direct comparison with costs developed by Zellars-Williams (1979) for an offshore southeastern U.S. deposit and updated as summarized in Table 9. Combined fixed and working capital requirements, at about $34 per annual ton of phosphate rock (Table 10), are approximately one-third lower than those estimated by Z-W (Table 9). The difference can be rationalized by noting that, in the Trondsen and Mead study, tug and barges used for transportation are assumed to be leased rather than purchased; onboard primary concentration (in this case, simply washing) is not explicitly costed; and flotation is not part of their beneficiation treatment. Updated direct operating costs are also about one-third lower than those estimated from Z-W figures for a 66% BPL concentrate.

Capital recovery charges would increase the break-even operating cost to about $63 million per year, or $16 per ton of product. If the product were to be sold at 80 percent of the current price of Florida and No. Carolina 66% BPL grade, revenues would be in the range $13-$14 per ton before applying the added (apparently 25%) discount proposed by Trondsen and Mead. The project clearly would not be attractive at current prices.

Installation of a full beneficiation sequence might (not guaranteed with an untried ore) increase concentrate grade significantly and, with it, selling price. However, overall recovery would be decreased and capital and operating costs would be higher as the result of the further processing. Conceptual optimization beyond the present preliminary level of evaluation would be built mainly on speculation.

Given the various features and uncertainties in the economic evaluations of offshore phosphorites, there appears to be no resultant basis for concluding that the aggregate southern California resource is inherently more or less attractive than the Georgia/South Carolina resource. In both cases, large uncertainties are associated with orebody definition, from the prospect of a major commercial interest, and with the specification, practical feasibility, and costs of the offshore operations.

Until information becomes available from further efforts in the directions just mentioned, progress toward definitive assessment of economic attractiveness will probably be quite limited. At the same time, preliminary economic evaluation of interesting, more recently explored phosphorite occurrences, as in Onslow Bay, North Carolina, may help in establishing priorities for the essential further work on delineation and on offshore mining systems and mineral processing operations.

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TABLE 1.—PARTIAL ANALYSIS OF A PHOSPHORITE CONCRETION (MURRAY, 1885) AND COMPARATIVE (MEAN) VALUES FOR SEVERAL PHOSPHORITES ON THE NORTHERN BLAKE PLATEAU (THIS STUDY; SEE TABLE 5 FOR A FULL LIST OF CHEMICAL CONSTITUENTS). (DATA FOR MURRAY CONCRETIONS: *S.S. BLAKE* STA. 317, 31°57′ N, 78°18.5′ W (610 m).)

	Murray (1885)	Phosphorites (This Report)
CaO	52.16	50.0
MgO	1.01	1.2
P_2O_5	23.5	22.2
CO_2	15.56	11.4
SO_3	2.29	1.0 (variable)
F	2.28	2.2
Cl	0.16	—
Insol. Residue	0.52	—
Ign. loss (includes CO_2)	18.71	14.9

From Manheim, Pratt and McFarlin (1980).

Note, in Table 1, that initial sampling of Blake Plateau phosphorite took place a full century ago.

Table 2. Offshore phosphorite analyses, Southern California

Chemical analyses for the 20 samples as received are given in Table 3.* The average analysis of the sample group is as follows.

P_2O_5	27.0 [BPL - 58.97 percent]
Insoluble	8.5
Fe_2O_3	2.3
Al_2O_3	0.66
CaO	43.0
CO_2	4.96
SO_3	1.89
F	3.10
LOI	10.15

Analyses of hand-picked nodules and pellets, individually tabulated in Table 4,* have an average grade as follows.

Chemical Composition	Percent	After Calcining
P_2O_5	28.2 [BPL - 61.59%]	31.4
Insoluble	6.63	7.4
Fe_2O_3	2.01	2.2
Al_2O_3	0.59	0.66
LOI	10.22	--

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* From Inderbitzen, Carsola and Everhart (1970)

TABLE 3.—SELECTED MINERAL PARAMETERS OF BLAKE PHOSPHORITE AND STANDARD FLORIDA PHOSPHATE ROCK. (CHEMICAL VALUES IN WEIGHT PERCENT. UNIT CELL (A) DIMENSION DETERMINED BY DANIEL APPLEMAN, SMITHSONIAN INSTITUTION)

	F	CO_2	P_2O_5	F/P_2O_5	Sp. Gravity	Index Refr.	A (Å)
Blake Plateau	2.5	11.4[a]	22.2	0.11	2.9 – 3.1	1.600	9.305
NBS STD 120b	3.84	2.79	34.57	0.111	3.0	1.600 –1.620[b]	—

[a] A portion of the CO_2 is due to physical admixture of calcite.

[b] National Bureau of Standards Phosphate Rock (Florida land pebble). Apatite in isotropic phase with considerable radiation damage has ω and ε = 1.62 (C. C. Woo, oral communication, 1978).

From Manheim, Pratt and McFarlin (1980)

Table 4. Chemical Analyses of Some Pellet Phosphorites*†

Location Code	P_2O_5	CaO	Fe_2O_3	Al_2O_3	MgO	Na_2O	K_2O	MnO	Cr_2O_3	TiO_2	V_2O_5	SiO_2	CO_2	SO_3	F	Cl	Org. C	Insol.
1	30.31	45.52	1.54	1.31	0.52	0.24	0.16	0.06	0.02	0.07	0.01	10.26	4.02	0.91	3.62	0.01	0.30	
2	34.42	49.34	1.20	0.95	0.17	0.22	0.12	0.02	0.01	0.16	0.01	4.21	3.12	0.88	3.84	0.01	0.18	
3	32.95	46.46	0.55	1.50	0.25	0.60	0.15					4.25	3.40	0.90	3.60		0.15	4.60
4	30.06	49.00	0.46	0.40	0.50										3.83		1.20	2.28
5	33.10	52.10	0.75	0.45	0.60										3.90		0.04	2.00
6	34.26	52.78	0.26	0.37	0.48	0.84	0.09					1.36	3.59	1.58	3.06	0.03		
7	37.21	54.24	0.13	0.39	0.10	0.27	0.06					0.97	2.64	0.62	4.17	tr		
8	33.06	51.96	0.21	0.43	0.40	0.80	0.07					2.46	4.48	1.22	2.49	0.03		
9‡	36.60	51.90	0.14	0.35	0.06	0.24	0.14					4.52	1.83	0.44	3.93	0.036	tr	
10	37.68	51.06	1.02	1.15	0.04	0.17	0.03					2.96	1.45	0.26	3.32	0.03		
11	36.85	51.69	1.30	1.00	0.03	0.27	0.05					2.99			3.76	0.12		
12	30.23	48.92	0.37	0.77	0.76	1.50	0.08					3.12	8.18	3.42	3.26	0.09		
13	34.58	54.25	0.45	0.46	0.77	0.60	0.02					2.35	1.25	1.60	4.00	tr		
14	29.86	48.96		1.50	0.98							4.96	6.09	2.51	2.96	0.40		
15	36.10	55.60	0.10	0.30	0.20	0.40	0.03					1.60	1.40	1.80	4.00	0.07		
16	31.20	47.70	0.36	0.27	0.35	0.58	0.06					3.00		1.16	3.00	0.23		
17	34.60	53.60										0.83			4.06	0.10		
18	34.50	42.50	1.76	3.50	0.40			0.40				9.60	0.95		3.06			
—19‡	29.00	46.14	2.15	1.96	0.77	0.73	0.32	0.03				6.78	5.33	1.61	2.96		0.33	
20‡	30.10	54.04	0.37	0.65	0.16				0.006		0.07	8.68	2.95		3.25		0.04	
21‡	31.30	43.86	2.10	3.05	0.10	0.34	0.46	0.11				12.66	1.76	0.52	4.06		0.07	
—22‡	29.90	47.04	3.56	1.31	0.91	1.46	0.35	0.01	0.06			3.70	5.41	3.22	3.49		1.04	
23‡	32.30	45.85	0.95	2.13	0.17	0.46	0.61	0.01	0.09	0.06	0.25	9.87	2.10	0.84	2.99		2.50	
24‡	33.12	47.70	0.51	1.26	0.30													
25‡	25.20	37.33	1.63	4.36	0.25	0.52	1.46	0.02	0.14	0.09	0.20	20.56	0.45	2.62				
26‡	33.00	47.86	0.75	1.28	0.36	0.81	0.77		0.25		0.39	6.06	2.16	1.86	3.28	2.83		

KEY TO LOCATION CODES

1. Central Florida (68% BPL grade from one producer)
2. Central Florida (75% BPL grade from one producer)
3. North Florida
4. North Carolina (Texasgulf Inc.)
5. North Carolina (Texasgulf Inc., calcined)
6. Morocco (Khouribga area, 75% TPL grade)
7. Morocco (Khouribga area, 80-82% TPL grade)
8. Morocco (Youssoufia)
9. Spanish Sahara (80% TPL grade)
10. Senegal (Taiba)
11. Togo
12. Tunisia (66-68% TPL grade)
13. Algeria (Djebel-Onk)
14. Arab Republic of Egypt (Kosseir area)
15. Israel (Oron)
16. Syria (Kneifees area)
17. Jordan (El Hassa)
18. North Vietnam (Lao-Kay)
—19. Baja California, Mexico (beach sands, beneficiated, not calcined)
20. Mexico (Saltillo area, La Caja formation, calcined, washed)
21. Tennessee, Maury Co. (washed brown rock)
—22. 40-mile bank off coast of Southern Calif. (sea floor nodule)
23. Idaho, Caribou Co. (Phosphoria Frm., main bed—high grade, 6-ft thick)
24. Idaho, Caribou Co. (washed, calcined)
25. Idaho, Caribou Co. (electric furnace shale rock—not beneficiated)
26. Tennessee, Maury Co. (washed electric furnace brown rock)

* Selected from Anon., 1971b.

† All are beneficiated concentrates except where otherwise indicated.

‡ Obtained from various sources other than *.

From G.D. Emigh (1983)

Table 5. Chemical analyses of impure marine carbonate fluorapatite concentrates and structural formulas of the apatites. (Only the constituents of apatite are reported.)

	1 Florida[1]	2 Morocco[1]	3 WUSA[1]	4 S-W Africa[2]	5 Peru[2]	6
			weight percent			
CaO	43.7	51.8	44.5	41.5	29.5	
MgO	.32	.37	.26	1.6	1.5	
Na_2O	.51	.73	.54	—	—	
P_2O_5	29.9	33.9	31.1	28.1	20.7	
CO_2	3.1	4.05	1.4	5.7	2.8	
F	3.5	4.2	3.4	2.7	2.1	
			Structural formulas			
	10 cations		6 anion groups			F anions

From Manheim and Gulbrandsen (1979)

Table 6. Posted prices*
as of July, 1986

Phosphates

Florida, land pebble, run of mine, dry basis, unground, bulk, ex-mine, average	Domestic	Export
60-66% BPL	$34.99	$30.36
66-70% BPL	$25.99	$34.67
70-72% BPL	$27.84	$37.38
72-74% BPL	$36.24	$41.80
74% BPL	$35.10	$50.20
Morocco, 75-77% BPL, FAS Casablanca		$48.50
70-72% BPL, FAS Casablanca		$46
Tunisia, 65-68% BPL, FAS Sfax		$32-38
Nauru, 83% BPL, long ton, FOB		—

* Industrial Minerals
(monthly journal)

Table 7. Mineral Content - Percent Weight

Mineral	1200 X 106 Microns	106 X 38 Microns
Quartz	50	33
Feldspar	14	14
Phosphate element	13	8
Igneous rock fragments	10	1
Heavy minerals *	8	41
Calcite	5	3
Total	100	100

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* The heavy mineral suite consists mainly of ferromagnesians (hornblende), with some magnitite and ilmenite and traces of zircon and epidote.

From Gruber, Raulerson and Farias (1986)

Table 8. Key to references and deposit identification.

[1] CC&P: Chambers Consultants and Planners (1979)

[2] Z-W: Zellars-Williams, Inc.(1979)

[3] Santo Domingo: G.A. Gruber, J.O. Raulerson and R.P. Farias (1986)

Table 8. Hypothetical offshore phosphorite production and actual beach phosphorite mining.

	CC&P[1]		Z-W[2]	Santo Domingo[3]
Geographical Location	Offshore Southern California		Offshore Tybee Island (Georgia)	Pacific Coast of Baja California Sur
Deposit Type	miscellaneous: 22-37% P_2O_5 nodules to 8% P_2O_5 sands		marine sediments: pebbles and sands	beach sand placer: largely unconsolidated
Ore Grade %P_2O_5	unspecified		11.1	3.2
Ore Grade %BPL	-do-		24.2	7.0
Overburden:	unspecified		4.0 ft, sandy	not discrete stratum
Matrix Depth:	-do-		23.0 ft	21-23 meters
Dredge Type	Case I drag-bucket	Case II hydraulic-suction	cutterhead suction dredge	(2) cutter-head dredges
Mining Rate st/yr	0.193×10^6	1.335×10^6	20×10^6 (incl. overburden)	23×10^6
Mining Rate st/hr	unspecified	unspecified	2,800	3,000
Phosphate Rock Prod'n Rate, st/yr	0.193×10^6	1.335×10^6	2.5×10^6 $\rightarrow 3.5 \times 10^6$	1.3×10^6
Mineral Beneficiation	crushing only, or crushing + calcining		crushing, washing, flotation, calcining	washing, flotation, gravity concentration
Product Grade %P_2O_5	32.7		≤ 30	29
Product Grade %BPL	71.5		≤ 66	63
Phosphate Recoveries mining:	no information		85%	51%
Phosphate Recoveries process:	-do-		66%	
Profitability	"negative NPV"	"very significant pos. NPV"	"could be economically feasible"	"economically feasible, but* "

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* ... sensitive to consumption and delivered cost of flotation reagents.

Table 8 Supplement. Hypothetical offshore phosphorite production.

	Trondsen and Mead	
Geographical Location	Offshore Southern California (8 deposits, 75-400 m avg.[a])	
Deposit Type	variable: phosphate nodules and phosphatic sands	
Ore Grade $\%P_2O_5$	25.5 (grade of nodules)	
Ore Grade %BPL	55.7	
Overburden:	unspecified	
Matrix Depth:	-do-	
Dredge Type	Case I hydraulic suction	Case II jet lift[b]
Mining Rate st/yr	1.5×10^6 or 6.0×10^6	
Mining Rate st/hr	375 or 1500 (2/3 nodules + 1/3 silt and sand)	
Phosphate Rock Prod'n Rate, st/yr	before processing: 1.0×10^6 or 4.0×10^6	
Mineral Beneficiation	crushing, screening and calcining	
Product Grade $\%P_2O_5$	ca. 29 (inferred)	
Product Grade %BPL	<66 (ca. 63?)	
Phosphate Recoveries mining:	no information[c]	
Phosphate Recoveries process:	no information	
Profitability (per original account)	unattractive (low rate) to marginal	attractive at both mining rates

---

[a] range of average water depth; assumed mining depth = 750 ft(230m)

[b] based on an innovative jet-flow alternator mining device of Continental Dredge and Marine Corporation; present state of development unknown.

[c] The same tonnage figures appear to have been used for dredged nodules and calcined concentrate.

Table 9. Revision, into current (1986) dollars, of cost figures reported by Zellars-Williams (1979) for 3.5 million ton per year, 68% BPL concentrate production from a hypothetical phosphorite deposit offshore Tybee Island, Georgia. $MM = million dollars.

Sector	Capital Costs $MM	Operating Costs	
		$MM/year	$/ton
Dredging and primary concentration	17.2	8.7	2.49
Transport to port	25.6	7.2	2.06
Processing to 66 BPL	80.3	21.6	6.17
Calcining to 68 BPL	32.7	12.3	3.51
Contingency	14.5	2.4	0.69
Working Capital	13.6	-	-
Total	184.	52.	14.9

Table 10. Revision, into current (1986) dollars, of cost figures reported by Trondsen and Mead (1977) for 4.0 million ton per year phosphate production from Southern California offshore phosphorite nodules; no offshore mineral processing. $MM = million dollars.

Sector	Capital Costs $MM	Operating Costs	
		$MM/year	$/ton
Dredging	6.1	4.5	1.13
Transport to port	-	6.2	1.55
Unloading	3.0	0.8	0.20
Crushing/Screening/ Calcining	102.3	25.1	6.27
Contingency	11.1	1.8	0.45
Working Capital	14.4	-	-
Total	137.	38.	9.6

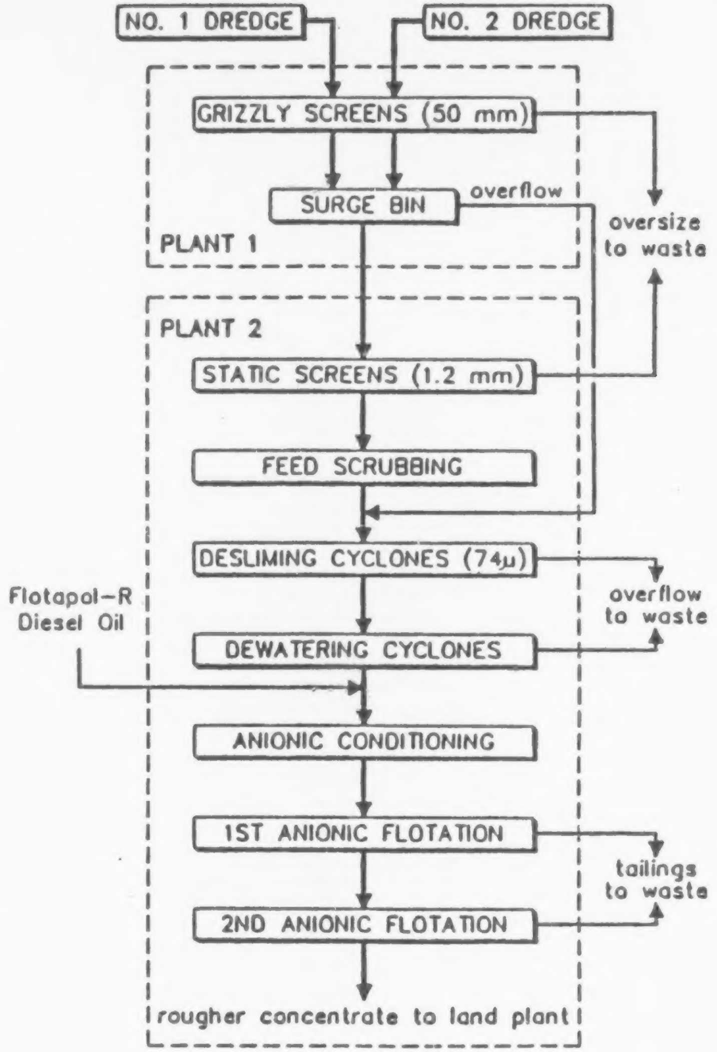
NO. 1 DREDGE
NO. 2 DREDGE
GRIZZLY SCREENS (50 mm)
SURGE BIN
overflow
PLANT 1
oversize to waste
PLANT 2
STATIC SCREENS (1.2 mm)
FEED SCRUBBING
DESLIMING CYCLONES (74μ)
Flotapol-R Diesel Oil
overflow to waste
DEWATERING CYCLONES
ANIONIC CONDITIONING
1ST ANIONIC FLOTATION
tailings to waste
2ND ANIONIC FLOTATION
rougher concentrate to land plant

Fig. 1a. Santo Domingo Floating Plant Block Flow Diagram

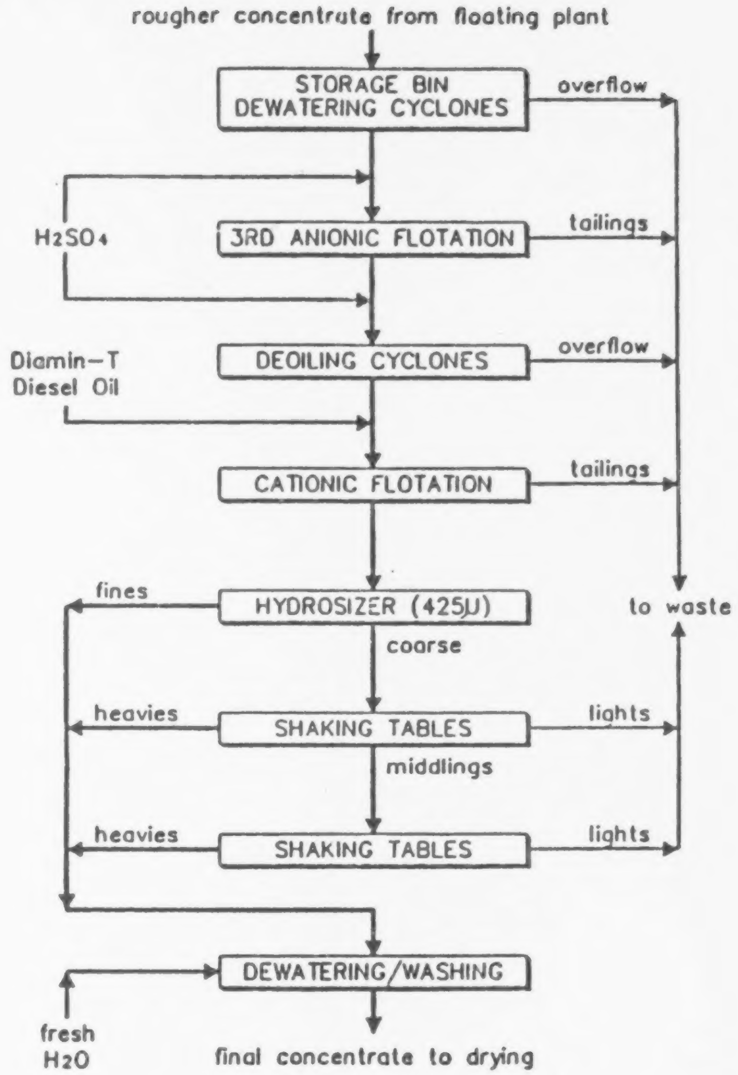
rougher concentrate from floating plant
STORAGE BIN DEWATERING CYCLONES
overflow
H2SO4
3RD ANIONIC FLOTATION
tailings
Diamin-T Diesel Oil
DEOILING CYCLONES
overflow
CATIONIC FLOTATION
tailings
fines
HYDROSIZER (425μ)
to waste
coarse
heavies
SHAKING TABLES
lights
middlings
heavies
SHAKING TABLES
lights
DEWATERING/WASHING
fresh H2O
final concentrate to drying

Fig. 1 b. Santo Domingo Land Plant Block Flow Diagram

Figures from Gruber, Raulerson and Farias (1986)

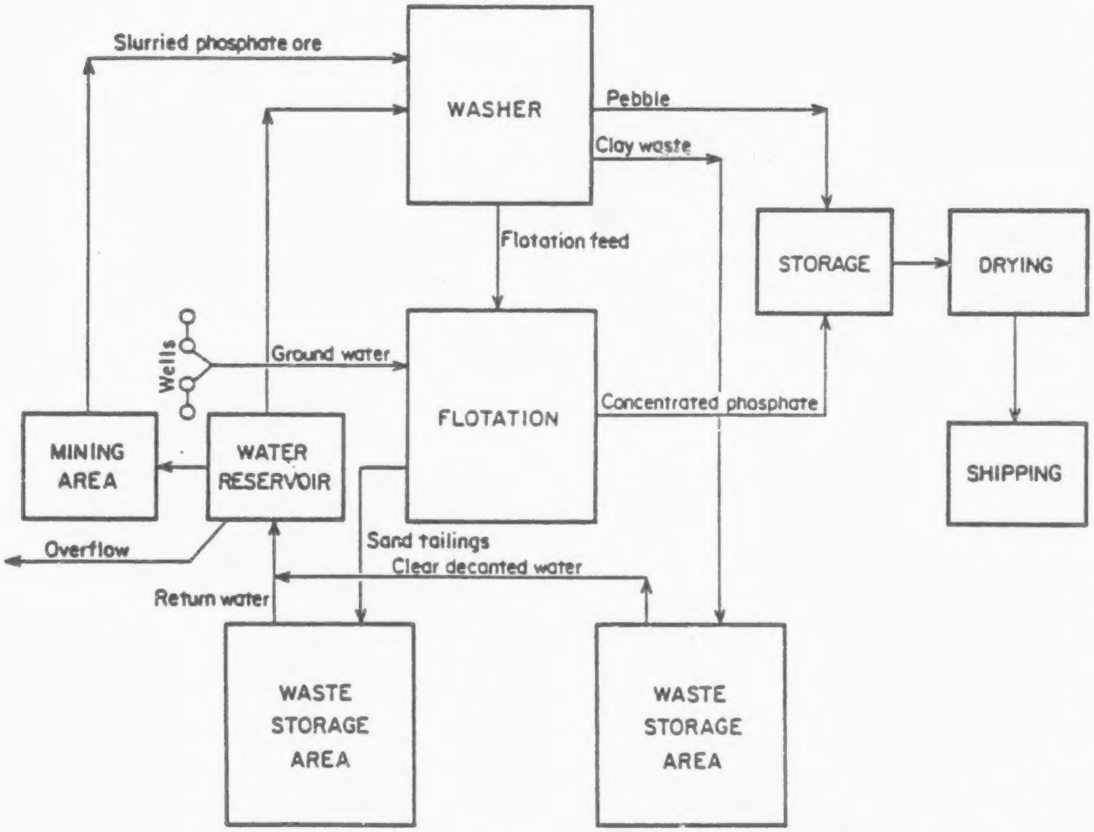
Slurried phosphate ore
WASHER
Pebble
Clay waste
Flotation feed
STORAGE
DRYING
Wells
Ground water
FLOTATION
Concentrated phosphate
SHIPPING
MINING AREA
WATER RESERVOIR
Overflow
Sand tailings
Clear decanted water
Return water
WASTE STORAGE AREA
WASTE STORAGE AREA

FIGURE 2. Typical process flowsheet, Southeastern United States.

(from Fantel, Sullivan and Peterson, 1983)

3. Processing Considerations for Offshore Chromite Heavy Metal Placers

Consulting and R & D in Extractive Metallurgy, Energy and Environmental Technologies

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PROCESSING CONSIDERATIONS FOR OFFSHORE CHROMITE HEAVY MINERAL PLACERS

W.W. Harvey and F.C. Brown

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D. Effect of New Technologies

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November, 1986

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PROCESSING CONSIDERATIONS FOR OFFSHORE CHROMITE HEAVY MINERAL PLACERS

A. Preliminary Notes

1. Synopsis of U.S. chromium industry

U.S. sources and uses of primary chromium are summarized in chart form on the following page. The sole economic chromium mineral is chromite, a substituted ferrous chromic oxide of high chemical durability.

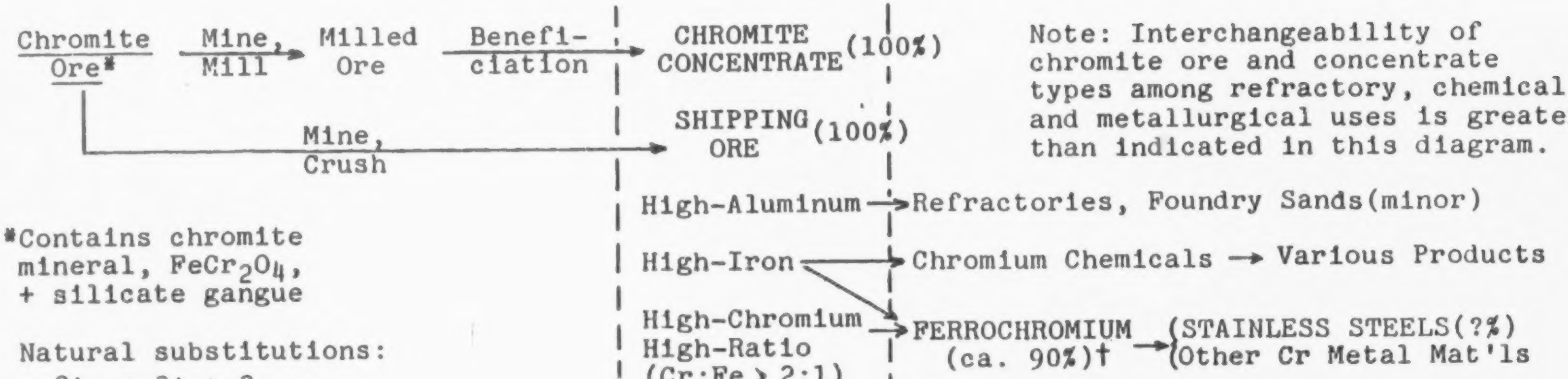
In brief, the U.S. imports substantially all of its chromite requirements for the production of chromium chemical intermediates and further products and for the production of chromite-containing refractories. Chromium chemicals demand has held relatively steady in recent years, while chromium refractories demand has been declining, owing to changing use patterns and to substitutions (Figures 1 and 2). Presently, at least 90-95% of chromite imported into the U.S. is for chemical and refractory uses.

It is no longer valid to imply, as do Figures 1 and 2, that import dependence is high for chromite used in the manufacture of ferrochromium, a class of high-Cr ferroalloys used in the manufacture of stainless steels and other chromium-containing alloys and metal materials. Rather, the U.S. has become almost entirely dependent upon imported ferrochromium (and other major ferroalloys) for conversion into manufactured goods (see Chart), and chromite demand for processing into metallurgical products has virtually disappeared.

The above pattern is well known by attendees at this Workshop, but the implications are not universally understood in the wider audience. The trends are very significant for consideration of potential offshore sources of chromite. Thus, rather than proposing to provide feedstocks for a domestic ferrochromium industry, one would, in reality, be proposing to produce a resurrection of that once vigorous industry. This could well be a worthwhile economic

Summary of U.S. Sources and Uses of Primary Chromium

Materials typed in capital letters are imported (approximate percentages in parentheses).



Chromite Ore*
Mine, Mill
Milled Ore
Benefi-ciation
CHROMITE CONCENTRATE (100%)
Mine, Crush
SHIPPING ORE (100%)
High-Aluminum
Refractories, Foundry Sands(minor)
High-Iron
Chromium Chemicals
Various Products
High-Chromium High-Ratio (Cr:Fe > 2:1)
FERROCHROMIUM (ca. 90%)†
STAINLESS STEELS(?%)
Other Cr Metal Mat'ls

Note: Interchangeability of chromite ore and concentrate types among refractory, chemical and metallurgical uses is greater than indicated in this diagram.

*Contains chromite mineral, $FeCr_2O_4$, + silicate gangue

Natural substitutions:

$Fe^{2+}, 2Cr^{3+}, 4O^{2-}$

$Mg^{2+} \rightarrow Fe^{2+}$; $Fe^{3+} \rightarrow Cr^{3+}$; $Al^{3+} \rightarrow Cr^{3+}$

†Greater, if conversion of 50,000 s.t. chromite in U.S. stockpile excluded.

1985 Dollar Values (Approximate)**

120,000 s.t. contained Cr in	415,000 s.t.	imported chromite:	$20 million
190,000	330,000	imported FeCr:	$160 million

Remarks: Not only dollar value, but also tonnage of chromium imported as ferrochromium now considerably exceed corresponding amounts for chromium imported as chromite; this does not include chromium contained in net imports of stainless steel products.

In other words, the U.S. has become more dependent upon imports of ferrochromium than of chromite. Demise of the domestic ferroalloy industry can be seen as a greater economic and strategic concern than lack of economic chromite ore resources.

** Quantities rounded from information provided by J. Papp (1986).

and strategic goal, but one that is not freely accomplishable in a market economy faced with enormous reserves of all grades of chromite in other parts of the world, coupled with insignificant economic resources of chromite here in North America.

A second major factor militating against revival of the domestic ferrochromium industry is the large production capacity already in place in the market economy countries. A low estimate (Dresler, 1986) is 2.4 Mt (million metric tons), as compared with an estimated current demand of 1.7 Mt (including ca. 0.36 Mt for the U.S. and ca. 0.03 Mt for Canada). Dresler points out that no shortage of supply of ferrochromium to world markets was encountered in the past.

Adding to the current excess of world production capacity, it is likely that the trend will continue toward vertical integration on the part of chromite mining companies, in foreign countries, into production of higher value ferrochromium. What is needed for this, essentially, are capital and abundant supplies of low-cost energy and raw materials for smelting. A further advantage in terms of potential savings in transportation costs, per unit of contained chromium, of product to market (%Cr in chromite concentrate or shipping ore is roughly half of %Cr in ferrochromium) is apparently not wholly realized in practice. This is because ferrochromium is more expensive to handle and ship, on an equal weight basis (Watson, 1986). As examples of value added, it is noted that Transvaal nominally 44% Cr_2O_3 ore at \$40-42 per tonne (metric ton) is worth about 6¢ per lb contained Cr (ca. 4¢ per lb contained Cr_2O_3); so-called "charge chrome", the largest tonneage ferrochromium traded, at 50-55% Cr, 6-8% C is worth about 40¢ per lb contained Cr (Table I).

Supply-demand comparisons convince Dresler (1986) that "no new chromite or ferrochromium producers should enter the world markets unless superior chromites or superior technologies are successful in competing with existing producers". Two qualifications may be made in terms of the present assessment of offshore chromite H.M. placers. First, the new source of chromite concentrate might be considered for manufacture of sodium dichromate, the major industrial chromium chemical and chemical intermediate, at a West Coast location. Not long ago, domestic capacity was substantially reduced by the closing of one of the three major chromium chemical plants. Alternatively, as a means of maintaining a larger ferrochromium production capacity than that being utilized in the relatively short-term conversion of 50,000 tons per annum of chromite in the national strategic stockpile, the offshore chromite concentrate could be used as feed to a ferrochromium plant (presumably a greenfield plant, at a West Coast location). The ferrochromium so produced would probably not be competitive for metallugical uses in normal markets but could be considered as intermediate for the production of chromium chemicals and derivative products, all of which have much higher values per unit of contained Cr than does high-carbon ferrochromium.

More thorough treatments of many of the above ideas and circumstances are contained in, among other sources, the informative chromium availability assessments (domestic - Lemons et al., 1982; foreign - Thomas and Boyle, Jr., 1984) and in various BuMines chromium chapters and surveys by J. Papp.

2. Mining and processing of the black sands

Griggs (1945) and Hornor (1918) should be consulted for fascinating accounts of mining and exploration activities directed toward the black sand placers of southern Oregon and northern California. In addition to their contents of gold and platinum, the black sands contain a number of heavy minerals, including chromite, ilmenite, magnetite, rutile, and zircon and other heavy silicates such as garnet.

Burns (1979) remarked that, during and after WW II, a cumulative approximately 10,000 tons of chromite concentrates was produced from raised terrace deposits on the Oregon coast. At one mine operated by the Humphrey's Gold Corp., chromite was recovered by gravity separation employing Humphreys spirals. This was the first commercial application of this high capacity and relatively low cost, gravity concentration device (Burt and Yashin, 1984).

The occurrence of heavy minerals, including chromite, in sand and gravel and in the tailings of placer gold and industrial mineral operations in the Pacific Northwest is qualitatively similar to that in the black sands. Recovery of by-product heavy minerals from these sources has been effected in the laboratory with use of combinations of beneficiation methods appropriate to the H.M. suite (Gomes et al., 1979; Martinez et al., 1981).

Figure 3 gives a partial flowsheet for the treatment of tailings from a former hydraulic mining operation. Magnetite was removed by low-intensity magnetic separation, and an ilmenite-chromite fraction was recovered by use of high-intensity magnetic and high-tension electrostatic separators. Since separate chromite and ilmenite grains normally have markedly different magnetic susceptibilities, ilmenite being the more strongly paramagnetic, the ilmenite-chromite sands could probably have been split magnetically, had that separation been an objective of the research.

The Figure 5 sequence includes preliminary H.M. concentration by spirals, gold recovery by amalgamation, and a separation of rutile from zircon. For the Washington State beach sand investigated, chromite was not present in sufficient quantity for its separation from the ilmenite to be of commercial interest. The approximate "split" of the chromite during beneficiation is shown in the figure. Analysis of the H.M. cleaner concentrate from the spirals was, in wt %: total Fe, 33; TiO_2, 22; ZrO_2(as zircon, $ZrSiO_4$), 8; Cr_2O_3, 1; and Au, 0.03 troy oz/ton (1.0 ppm). For comparison with these laboratory examples for beneficiating chromiferrous sands, a simplified flowsheet typical of a chromite ore concentrator is shown in Figure 4 from Aplan (1985). The material treated is fines resulting from the mining and handling of hard rock, so-called 'lump', chromite ore.

3. Production potential of offshore chromite placers

Griggs (1945) compared the beach and offshore deposits of southern Oregon, noting them to be similar in structure. Both consist of "many discontinuous layers containing quartz and heavy mineral grains in various proportions", from almost pure quartz sand to almost entirely heavy minerals. Stratum thickness, according to Griggs, ranges from a fraction of an inch to a foot or more, most strata not exceeding a few inches in thickness. Burns (1979) noted, simply, that offshore drilling in 1970 revealed some localized H.M. enrichment in near-surface sediments.

Overall, at least until recently, most disinterested observers have not been sanguine about the prospects for commercial chromite mining, either of the beach sands or of the lesser explored offshore placer deposits. For instance, the recent summary report on chromium (DeYoung, Jr. et al., 1984) states that there are no identified chromite placer deposits important enough, in terms of probable near-future production, to be included in the five-nation cooperative stategic minerals inventory.

Obviously, there are members of the optimistic camp; otherwise, the current evaluation would not be in progress. The question is how one arrives at even tentative conclusions, absent detailed information that would come from pre-development exploration. One approach is to consider consequences of the gross features of the postulated resource and the in-place receiver industries and infrastructure.

From the limited information available, offshore chromite H.M. sands appear unlikely to yield chromite concentrates that are high grade by world standards. See Table II for analyses of some commercial chromites, including refractory grade chromite from the Philippines. Not so many years ago, even pure mineral chromite would not have been judged of adequate grade for metallurgical uses because, although Cr_2O_3 content is high, the Cr:Fe ratio is relatively low ($FeCr_2O_4$: 68% Cr_2O_3; Cr/Fe = 0.93). Today, however, high Cr/Fe ratio is not as important. Charge chrome (high-carbon ferrochromium, 50-55% Cr, averaging 53% Cr), which accounts for over 70% of ferrochromium usage in industry, is commonly made from 1:1 ratio ores (Watson, 1986). On this revised basis, chromite concentrates from offshore heavy mineral placers might be suitable, in terms of their composition, for metallurgical as well as for chemical use. Agglomeration would, however, be required for conventional smelting to ferrochromium (Figures 6 and 7).

Next, transportation costs would generally be unfavorable for chromite production in the Pacific Northwest, because of concentration of ferrochromium plants (whether shut down or operating intermittently or at reduced capacity) and chromium chemicals and refractories producers in the eastern half of the country. It may be noted that the potential exists for conversion from one type of ferroalloy plant to another, e.g., ferromanganese to ferrochromium, but this possibility does not change the broad issue of transportation costs. The economic consequences of having to move concentrates to distant plants are sufficient to warrant consideration of a new metallurgical, chemicals or refractories plant, as appropriate.

One can foresee close parallels in the mining and beneficiation of chromite and titanium H.M. placers. If the comparison is valid, for a strategy whereby a chromite H.M. primary concentrate is produced offshore in conjunction with the dredging operation, the cost structures should also be roughly comparable. It will have been noted that the value ranges of chromite concentrate and ilmenite concentrate overlap, depending upon grades. These considerations suggest that chromite alone would not meet revenue requirements, but that higher value minerals must also be present and recoverable. Since, for economic feasibility, additional revenues from the sale of supplemental minerals must exceed the costs of their separation and concentration, they would need to be present in at least coproduct amounts in the ore.

To illustrate the point, if minerals of comparable or lesser value, such as ilmenite and magnetite, respectively, were present in recoverable and salable quantities, their recovery would not be as advantageous as the presence of equal, incremental amounts of chromite. If precious metals were present in really significant quantities, such that their value as concentrates would exceed the cost of mining the H.M. placer and beneficiating the gold and platinum, the enterprise might elect to discard the chromite-bearing tailings.

In view of the appreciable costs predicted for dredging and offshore primary concentration of titanium H.M. marine placers (Mining chapter), there may be an economic advantage in making the H.M. concentrate at a shore plant. A local market for the sand tailings would be helpful in this regard. A differently oriented, further appraisal would be required to estimate the effect of this change on the economics.

B. Costs for a S.W. Oregon Beach Sands Placer

In the 1982 domestic chromium availability appraisal (BuMines IC 8895), Lemons et al. costed a strip mining and beneficiation operation for producing chromite concentrate from southwest Oregon beach sands. Ore grade was taken to be 5.6% Cr_2O_3, from which a high-iron chromite concentrate would be produced by a typical H.M. sands beneficiation flowsheet (gravity, magnetic and electrostatic concentration steps). Estimated costs were based on gravity and electrostatic separation.

Annual mining capacity was considered to be 1×10^6 tonnes of ore, at which rate the demonstrated resource would be depleted in about 11 years of mining. Concentrate grade was taken to be 42% Cr_2O_3, with Cr:Fe = 1.5:1. From the given ore and concentrate grades and an assumed 80% chromite recovery in beneficiation, mill production is estimated at about 107,000 tonnes of concentrate per year. Thayer (1973) had previously reported a concentrate grade of about 40% and 1.6:1 for S.W. Oregon coast chromite, from a review of earlier work.

At the indicated grade, the concentrate could probably be used for the production of chromate chemical or the charge chrome type of high-carbon ferrochromium. The price advantage of chromite meeting refractories specifications (Tables I and II) could probably not be realized; the concentrates do not appear to be of the high-aluminum, high total Al_2O_3 plus Cr_2O_3 variety and might not meet other specifications for refractory grade chromite.

Lemons et al.(IC 8895) estimated the fixed capital requirements (in 1981 dollars) to be \$4-\$6 per annual tonne (metric ton) of ore for the mining operation and \$13-\$14 per annual tonne for the mill. Mine and mill cash operating costs were estimated by them to be \$1-\$2 and \$4-\$5, respectively, per tonne of ore. These costs were based on 300 days of operation per year (82% operating factor). From those costs, and with use of the evaluation methodology of the minerals availability system, the IC 8895 authors estimated that revenues of about \$180 per tonne (say, \$160-\$165 per short ton) of chromite concentrate would be required for profitable operation (15% DCFROR).

The foregoing cost figures have been updated to current (1986) dollars, as summarized in Table III for the production of 120,000 tons (short tons, s.t.) per year of chromite concentrate from southwest Oregon beach sands. The corresponding mining rate would be 1.1×10^6 s.t./yr. Examples of the cost-updating methodology employed are given in the next section of this chapter, as well as in the Appendix of the Titanium H.M. chapter.

For the updated, estimated fixed capital cost of $\$22._6$ million (Table III), depreciation expense (5-year, straight line) is about $\$37._7$ per ton, so that break-even operating cost (i.e., cash cost plus depreciation) is approximately $100 per ton. The requisite cash flow (i.e., after-tax profit plus depreciation) for a five-year payback would be reached, under current tax law, at a selling price of about $160 per ton of concentrate. For a payback period of three years, which would be more appropriate for a higher risk operation entailing mining and some degree of beneficiation offshore, the revenue requirement would be increased to approximately $250 per ton.

By comparison with the above figures, the average price of low grade (>40%, <46% Cr_2O_3) nonrefractory chromite concentrate imported into the U.S. (predominantly from the Republic of South Africa) in the first half of 1986 was $40 per ton. This average price is exclusive of import duties, freight, insurance and other charges incurred in bringing the chromite to the U.S. (Papp, 1986). With respect to incentive for utilization of domestic chromite in ferrochromium production, G. Watson (OTA Workshop, September 18) suggested that a price of $40 per ton delivered to the Pittsburgh area would be of interest.

The approximately 1 million annual tons mining rate employed in the foregoing availability appraisal of chromite beach sands is resource-limited. Profitability would be improved by higher mining rate and longer mine life. This would require a much larger resource of chromite beach sands than has been demonstrated up to now.

Cost figures shown in the Offshore Titanium Placers chapter for a Ti/assoc. H.M. beach sands operation give an indication of the effect of a large incremental capital investment on the profitability of offshore chromite placer mining. Purchase, in 1980, of an unprofitable Ti/assoc. H.M. mine and mill entailed a fixed capital cost of about $12 MM (million). Together with current cash operating costs of $11.1 MM/yr, depreciation, at ca. $2.4 MM/yr, result in a total operating cost of $13.5 MM/yr. Revenues, for 145,000 annual (short) tons of separate H.M. concentrates, are currently running at $17.95 MM/yr, so that pre-tax profit is $4.45 MM/yr and after-tax profit, about $2.22 MM/yr. Improvements in plant and operation leading to current profitability entailed an average outlay of about $2 MM/yr in working capital for an unspecified period between one and three years (say, 2 years). Thus, the appropriate total investment on which to base an estimated return is about $16 MM, from which ROI = 2.22/16, or 14 %, and the payback is 3.5 years.

Present start-up capital costs, i.e., for a greenfield plant, are estimated to be $50-60 MM. By coincidence, this range includes the $57 MM total capital cost estimated by Horton (companion study) for mining, shipboard primary concentration, and dock facilities for 5,000 (short) tons per day of partially concentrated chromite sands. Depreciation cost for the indicated capital investment of $50-60 MM would be $10-12 MM/yr, giving a total operating cost of about $22 MM/yr and, consequently, negative pre-tax profit. In other words, that magnitude of required investment would render the present Ti/assoc. H.M. sands operation uneconomic. The implication for a new chromite beach sands or offshore placer operation is obvious. In the next section, a 1972 economic evaluation of an offshore chromiferous H.M. placer is considered and selectively updated for mining rates up to 19 million tons per year.

C. Costs for an Offshore Chromite H.M. Placer

A published summary of an economic assessment by Wilcox, Mead and Sorensen (1972) has some informative numbers and conclusions. In their NOAA-sponsored Sea Grant study, the authors assumed a H.M. offshore placer deposit containing chromite 5%, ilmenite 5%, magnetite 10%, gold 10 ppb (parts per billion) and platinum 0.23 ppb, i.e., 20% recoverable H.M. and traces of gold and platinum. The identified offshore location was an area defined by the USGS as comprising about 400 km^2 on the continental shelf between Crescent City, California and the Oregon border. The location was further identified by the authors as being mostly in state waters, although this is difficult to reconcile with the approximately 30-35 km distance from offshore Crescent City to the Oregon border.

Mining by bucket-line dredge was assumed to proceed around the clock for either 170 days (7 months of the year at an 80% operating factor) or 291 days (12 months at an 80% operating factor). The corresponding hours on line are approximately 4080 and 7000, respectively. In the Wilcox et al. scenario, a combination of tugs and barges transported the H.M. sands to a port facility, significant capital costs for the port being included explicitly in their economic analysis. A small support vessel was included in the projected fleet.

Wilcox et al. (1972) did not provide specifics of the sequence of mineral processing steps from dredged ore to products (Table IV). However, they indicated that an initial stage of beneficiation, carried out at sea, would separate and reject sand and gravel from the heavy minerals, to the extent of half of the volume of materials dredged. The cost of this partial separation step was given as \$0.20 (in 1972 dollars) per ton (s.t.) of dredged ore. A second stage of separation, on land, of the approximately 40% H.M. material transported to shore was estimated to cost \$0.50 per ton of dredged ore. Final processing of the H.M. concentrate was calculated to add \$1.30 per ton, yielding an estimated total cost of \$2.00 (1972) per ton of dredged ore.

Three bucket-line dredge sizes and corresponding mining rates were considered by Wilcox et al., namely: 9 cu ft (450 s.t./hr); 18 (900); and 54 (2700). In combination with the alternative running times of 170 days (4100 hr) and 291 days (7000 hr), six modes of operation were costed. Revenue projections indicated that cash costs would exceed revenues for all of the throughput rates considered (1.8 to 19 million tons per year mined ore). Costs presented in the 1972 paper have been updated in order to arrive at current revenue requirements for the highest-throughput case, where maximum cost advantage is derived from economies of scale.

Estimated fixed capital requirements were adjusted by use of the Marshall and Swift All-Industry Equipment Cost Index. In 1972, the index was 332. It is currently about 795, so that capital costs have increased on average by a factor of 2.4. All capital-related components of operating cost, such as dredge maintenance, tug and dock operation and maintenance, taxes and insurance, and depreciation would have increased by substantially the same factor of 2.4.

Labor costs in the 1972 paper were based on an annual compensation cost of $12 thousand; the currently appropriate cost is $28 thousand. Power costs were not given but are assumed to have been about 1.25¢/kWh, vs about 4¢/kWh presently, in the relatively low-cost power area of the Pacific Northwest. Finally, the cost of general materials and supplies is estimated to have escalated by a factor of 2.5 between 1972 and the present.

No breakdown of overall mineral processing cost was given by Wilcox et al. (1972), even though beneficiation cost was by far the largest component of their estimated total operating cost. Based on a typical titanium H.M. beneficiation plant (see Titanium H.M. chapter and references therein), the following distribution of the total cash cost has been assumed:

Materials and supplies, plus capital-related charges	20%
Utilities and fuels	25%
Labor	55%

With these major cost components each escalated by the previously specified factors, total beneficiation costs would have increased by a factor of 2.57.

In terms of throughput (in this context, quantity of products in relation to quantity of material processed), the breakdown of operating costs for beneficiation of the contemplated 20% H.M. ore could possibly be closer to that for a phosphorite beneficiation plant (see Phosphorite chapter and discussion therein of the Zellars-Williams cost analysis). The corresponding breakdown from Z-W is: materials & supplies, 30%; utilities and fuels, 15%; labor, 35%; and capital-related charges, 20% of total beneficiation costs in the land plant. With this alternative cost breakdown, total beneficiation costs would have increased by a factor of 2.53. The difference between this quantity and 2.57 (above) is well within the accuracy of the updating methodology and suggests that the lack of provision of a specified breakdown is not serious in this case. This is not to say that the estimated costs of beneficiation may be considered to have been substantiated in any way.

Estimated current costs for the Wilcox et al. scenario are summarized in Table V. Process plant capital requirements and operating cost amount to 50% and 70%, respectively, of total venture costs. A capital requirement for exploration and land acquisition, being nondepreciable, has not been carried over from the original Wilcox et al. tabulation. Table V includes an allowance for working capital.

With updated total fixed capital cost at $242 million, for 2.15 million annual tons of products (Table IV), depreciation expense is almost $23 per ton of products. Updated total cash operating costs are $64 per ton, so that the breakeven revenue requirement is approximately $87 per ton of products.

A summary of assumed product recoveries and revenues at current prices is given in Table IV. In this compilation, chromite recovery has been decreased from an unattainable 100% to a more realistic, but still optimistic 80%, and ilmenite recovery has similarly been decreased to 80% (from 89%). Other recoveries are those implicit in Wilcox et al. (1972). It is to be noted that, with the subsequent order-of-magnitude increase in the price of gold (the authors used a 1972 free-world market price of $58 per troy ounce), revenues from sale of the approximately 10 ppb gold content would now contribute 2% or so to total revenues. Revenues from the 0.23 ppb platinum content would remain relatively inconsequential.

A comparison of estimated costs and projected revenues shows that the latter are clearly inadequate to justify the venture. This would be the case even if beneficiation plant operating costs were decreased from the tabulated $45.6 per ton of products to as low as $10 per ton. Revenues of almost $125 per ton would be required to realize a three-year payback on investment, under current tax law.

As is the case for the other offshore mining ventures considered in the present assessment of minerals of the EEZ, the project described by Wilcox et al. is highly capital-intensive. Re-estimated total investment is roughly $125 per annual ton of capacity for the maximum production rate considered in the 1972 evaluation, namely, 19 million tons per year dredged and 2.15 million tons per year of products. Unit costs would increase with decreased production rate and possibly exceed the $340 per annual ton investment cost for beach sand chromite (previous section) at the same low capacity of 0.12 million ton of product (Table III).

If the gold content were increased 20-fold (to 200 ppb, or 0.006 troy oz/ton ore), and the nominally 97% gold recovery (realistic?) maintained, revenues from the sale of gold would become largest among the five products. It would be interesting to estimate at what gold content it would become more economic to recover only the precious metals. The thinking is that elimination of the proposed mineral processing plant on land might more than compensate for the costs of adding and operating an amalgamation plant for precious metals recovery, in conjunction with the dredging operation, as described by Richardson and Horton (companion study).

D. Effect of New Technologies

The focus here is on those developments that might affect the utilizability and, more broadly, the economic feasibility of offshore chromite H.M. placers.

First, we consider developments related to the physical state of the chromite. Its presumed occurrence as largely unconsolidated sand of marine placer origin is favorable for separation and recovery by conventional heavy mineral sands techniques, as described here and in the section on offshore titanium H.M. placers. However, chromite sand concentrates, and chromite fines generally, are not suitable for direct feeding to a ferrochromium smelter, but are normally briquetted or otherwise agglomerated before feeding to the electric furnace (Figure 7). Related to this, an energy conserving technology has been introduced at some locations, whereby prereduction is effected by sintering chromite briquettes with carbonaceous reductant in advance of electric furnace smelting (Mikami, 1983).

In-flight plasma reduction is a developmental smelting technology that moves in the same direction of utilizing smaller diameter chromite particles, rather than lumps or agglomates. A number of approaches to plasma reduction of low-grade chromite have been investigated, including the shock-wave technique of Tylko et al. (1981). Electrical energy costs per unit of production appear to be very high, so that it is doubtful that plasma reduction will soon come into wide use in the smelting of low value oxide materials such as chromite concentrate.

Production of chromate/dichromate and of chromite refractories, by contrast with manufacture of ferrochromium, does not require lump ore or agglomerates. Rather, the chromite is typically ground to specification, as illustrated in the accompanying flowsheet for the manufacture of sodium dichromate (Figure 6). The diagram has been simplified by omission of an aluminum removal step. Caustic roast temperature here is relatively high, about 1200 °C, so that fuel requirements are substantial. An investigation of a lower temperature caustic roast and water leach treatment has been reported (Chandra et al., 1981) for the extraction of chromium from low grade domestic chromites.

Ferrochromium could be more widely used in place of chromite as feedstock for the production of major chromium chemicals and derived products. As an example of the possibilities, electrolytic chromium metal (as distinct from chrome plate) is presently made by electrolysis, after purification, of chromium-alum solution obtained by dissolution of ferrochromium (Papp, 1983). A possible effect of the indicated change in feedstock and technology on the utilizability of offshore chromite was suggested in Section A.

The interchangeability of various grades of chromite, particularly with respect to ferrochromium and chromate/dichromate production, but also including manufacture of refractories, has been noted. For the metallurgical industry, this flexibility is mainly the result of the advent of effective decarburization technologies in the production of stainless steels and other chromium-containing specialty steels and alloys. The most widely used of these methods is argon-oxygen decarburization (AOD); a second method in use is vacuum-oxygen decarburization. AOD and VOD make possible the utilization of cheaper high-carbon (and lower chromium) ferroalloys, that can be made from lower grade chromite than is required for the formerly specified low-carbon ferrochromiums. There are still uses that require the higher specification ferrochromium, but these account for less than 25% of total demand.

Further development of direct stainless steel production technology would improve the relative attractiveness of ferrochromium production from offshore chromite. By one approach, stainless steel ($\geq$ ca. 14% Cr) is produced by smelting together a high-iron chromite with iron ore. One can speculate further that the commonest stainless steel compositions containing nickel as well as chromium, such as 18Cr-8Ni, could be made by including ferronickel from the nearby Hanna nickel smelter.

Related experience of Rustless Iron and Steel suggests that a technological breakthrough would be required for direct stainless steel production. During 1940-1950, that company made stainless from chromite ore, steel scrap and ferrosilicon reductant in a conventional electric steelmaking furnace. However, quality was not the best, and overall energy input produced no savings over the older method of stainless steel manufacture by addition of ferroalloys (Watson, 1986).

Finally, we note again that location of the port facility and the mill at or close to a growing metropolitan region or another area of high construction activity could be advantageous. The main condition is that the tailings generated be suitable and in demand as gravel and/or cement and mortar sand. Then, the sale of the tailings, comprising 80-90% or more of the ore, might pay for the mining.

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Table I. Prices for chromite ore, ferrochromium and chromium metal, E/MJ, August 1986 and Industrial Minerals, July 1986 (lower left).

-Selectively revised Oct.'86-

ORES & CONCENTRATES

Tons of 2000 lb or units of 20 lb unless otherwise stated. Short ton unit—stu; long ton unit—ltu. (a) c.i.f. US ports, (b) f.o.b. ship pt, (c) f.o.b. mine or mill, (g) depending on grade, (i) import duty extra, (n) nominal, (t) term contracts.

Chrome Ore: dry basis, subject to penalties if guarantees are not met, f.o.b. loading point; term contracts (subject to negotiation) are generally lower.

Transvaal, mt	Revised:	$48-52
44% Cr_2O_3, no ratio		$40-42
Turkish, mt	Revised:	$110
48% Cr_2O_3, 3:1		$125
Russian, mt		
53-55% Cr_2O_3, 4:1 ratio (pricing basis 48% Cr_2O_3, 4:1 ratio)		Suspended

Chromite

Transvaal, chemical grade, 44/45% Cr_2O_3, FOB	$40-50
Transvaal, foundry grade, 45% Cr_2O_3, FOB	$45-55
Transvaal, refractory grade, 46% Cr_2O_3, FOB	$65-75
Philippine, refractory grade, concentrates, FOB	$100-120
Sand, moulding grade, 98% <30 mesh, del UK	£92-110

FERROALLOYS

(a) carload lots; (b) delivered; (c) lump, bulk; (d) f.o.b. shipping point; (n) nominal. Revised

Ferrochrome: lb contained Cr, (a), (c), (d)

Charge chrome(50-55%Cr,6-8%C)	40¢
High carbon(65-70%Cr,5-6.5%C)	52¢
Low carbon(67-73%Cr,.04-.05%C)	80¢

MISCELLANEOUS METALS

Wholesale lots. (a) f.o.b. ship pt.; (b) delivered; (c) f.o.b. N.Y.; (d) dep. on size of lot; (e) dep. on grade; (n) nom.

Chromium: lb of material, (a), eff. 6-1-86

electrolytic, 99.8%	$3.00-3.75
vac. melting, 99.5%, eff. 10-11-85	$4.62

Table II. **Analyses of Typical Chromites and Corresponding Chrome Ores**

	No. 1		No. 2		No. 3		No. 4		No. 5		No. 6	
	C*	O*	C*	O*	C*	O*	C*	O*	C*	O*	C*	O*
Cr_2O_3	47.41	44.52	36.24	32.10	61.44	54.91	57.70	47.00	58.32	48.50	55.90	49.19
Al_2O_3	14.82	15.50	31.86	30.20	11.41	9.92	13.44	12.65	11.06	10.01	11.92	11.37
Fe_2O_3	9.21		2.97		nil		3.42		4.10		2.18	
FeO	16.86		11.32		12.53		11.66		11.10		18.80	
Tot. Fe as FeO†	(25.26)	24.72	(13.99)	12.72	–	12.41	(14.70)	11.93	(14.75)	13.28		18.27
MgO	11.40	10.10	17.10	18.06	13.66	14.92	13.29	15.46	14.23	18.83	10.36	12.35
CaO		0.30		0.44		0.70		1.77		0.40		0.48
SiO_2		2.24		5.00		5.02		5.71		6.94		6.58
TiO_2	0.41	0.43	0.38	0.30	0.17	0.20	0.39	0.32	0.06	0.05	0.76	0.67
MnO	0.09	0.07	0.11	0.08	0.16	0.14	0.24	0.06	0.15	0.12	0.30	0.26
NiO			0.12	0.10					0.20	0.18	0.09	0.08
V_2O_5	0.33	0.30							0.07	0.05	0.18	0.16
L.O.I.				0.35		0.92		3.95		1.20		
Total	100.53	98.17	100.10	99.35	99.37	98.64	100.24	98.85	99.29	99.56	100.49	100.44
Cr:Fe	1.67	1.58	2.30	2.22	4.31	3.90	3.45	3.47	3.45	3.25	2.37	2.37

*C = chromite, O = ore or concentrate.

†Separate FeO and Fe_2O_3 are generally not determined. Generally only Cr_2O_3, Total Fe, Al_2O_3, MgO, SiO_2, and CaO are available in out-turn analyses. These are given on a dried basis. CaO and SiO_2 ranged from nil to 0.17 in separated chromites and were omitted.

No. 1—Transvaal Steelport. 2—Phillippine Masinloc. 3—Kempirsai (USSR). 4—Selukwe, Zimbabwe (talc-dolomite gangue). 5—Turkish. 6—Great Dyke.

(From Mikami, 1983)

Table III. Breakdown, by major sector, of capital and operating costs for production of 120,000 s.t./yr chromite concentrate from Southwest Oregon beach sands: update of IC 8895 costs (reported in January 1981 dollars). $MM = million current U.S. dollars.

Sector	Capital Requirement $MM	Cash Operating Cost $MM/yr	Cash Operating Cost $/s.t.
Preproduction Expenses	2.6	---	----
Mine and Equipment	6.0	1.9	15.8
Mill	16.6	5.6	46.7
Working Capital	15.8	---	----
Total	41.0	7.5	62.5

Per annual ton: $342.

Table IV. Production profile (slightly revised from Wilcox, Mead and Sorensen, 1972) and updated revenue estimates for production of 2,150,000 s.t./yr chromite and other H.M. from a California/Oregon offshore placer. $MM = million current U.S. dollars.

Product	Per Cent Recovery	Tons/Year Recovered	Sale Price $/s. ton	Revenue $MM/yr
Chromite	80	760,000	40	30.4
Ilmenite	80	760,000	45	34.2
Magnetite	33	630,000	42.5[a]	26.8
Gold	97	5,370[b]	400[c]	2.15
Platinum	98	125[b]	500[c]	0.06
Totals		2.15×10^6		93.6 ($43.5 per ton)

---

[a] At 3.8¢ per lb contained iron
[b] Troy ounces per year
[c] Dollars per troy ounce

Table V. Breakdown, by major sector, of capital and operating costs for production of chromite and other H.M. from a California/Oregon offshore placer; mining rate 19 million s.t./yr; update of Wilcox, Mead and Sorensen 1972 costs. $MM = million current U.S. dollars.

Sector	Capital Requirement $MM	Cash Operating Cost $MM/yr	Cash Operating Cost $/s. ton [a]
Dredge and Support Vessel	77.5	20.9	9.7
Tug and Barges	8.9	1.6	0.7
Dock	24.0	4.7	2.2
Process Plant	132.0		
H.M. concentration		27.2	12.7
Final separation		70.6	32.9
Working Capital [b]	24.2	----	----
General and Administrative Expenses [c]	----	12.5	5.8
Total	267.	137.5	64.
Per annual ton:	$124		

---

[a] Of products, based on 2.15 x 10^6 s.t./yr at 57% overall recovery.
[b] At 10% of fixed capital.
[c] At 10% of direct cash costs.

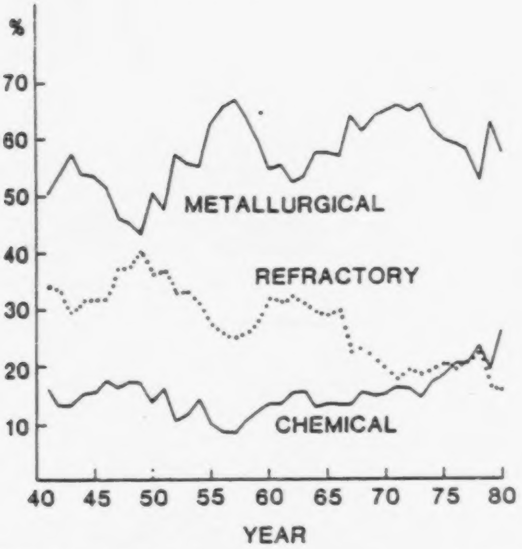


FIG. 1—*Percent usage of chrome ores by type in the United States, 1941–1980.*

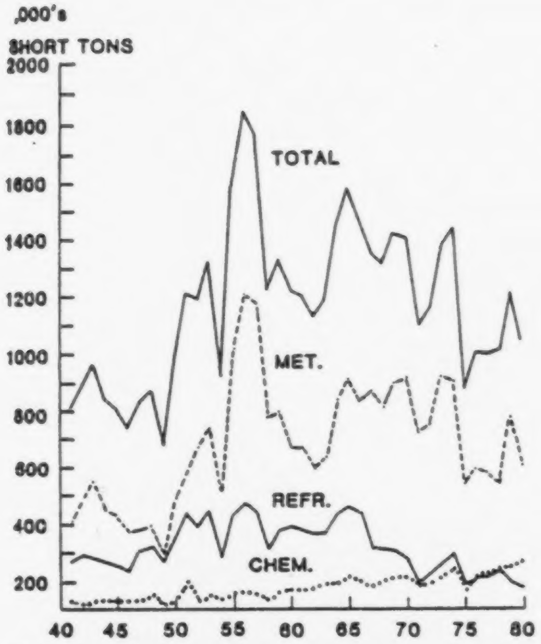


FIG. 2—*Consumption of chrome ores, total and by grade, in the United States, 1941–1980.*

(Above figures from Mikami, 1983)

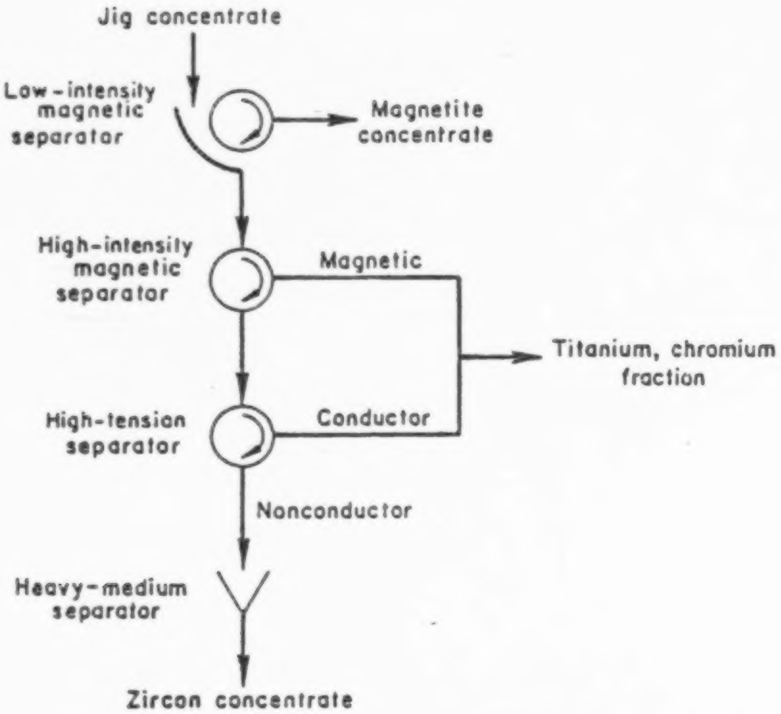

Jig concentrate
Low-intensity magnetic separator
Magnetite concentrate
High-intensity magnetic separator
Magnetic
Titanium, chromium fraction
High-tension separator
Conductor
Nonconductor
Heavy-medium separator
Zircon concentrate

FIGURE 3. - Sequence for processing Cherokee tailings.

(From Gomes, Martinez and Wong, 1979)

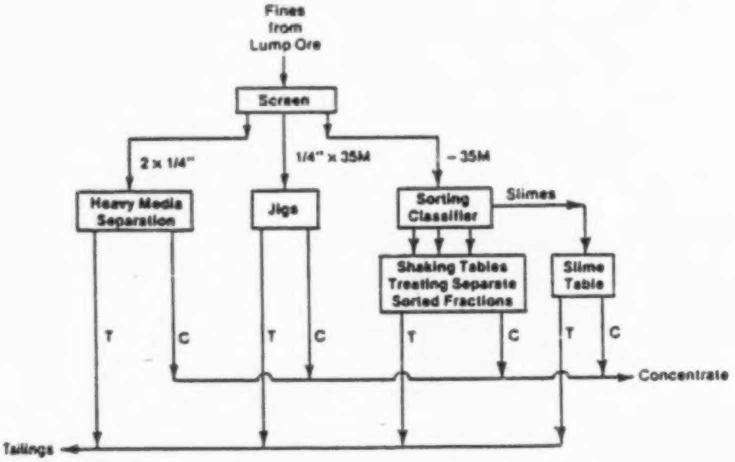
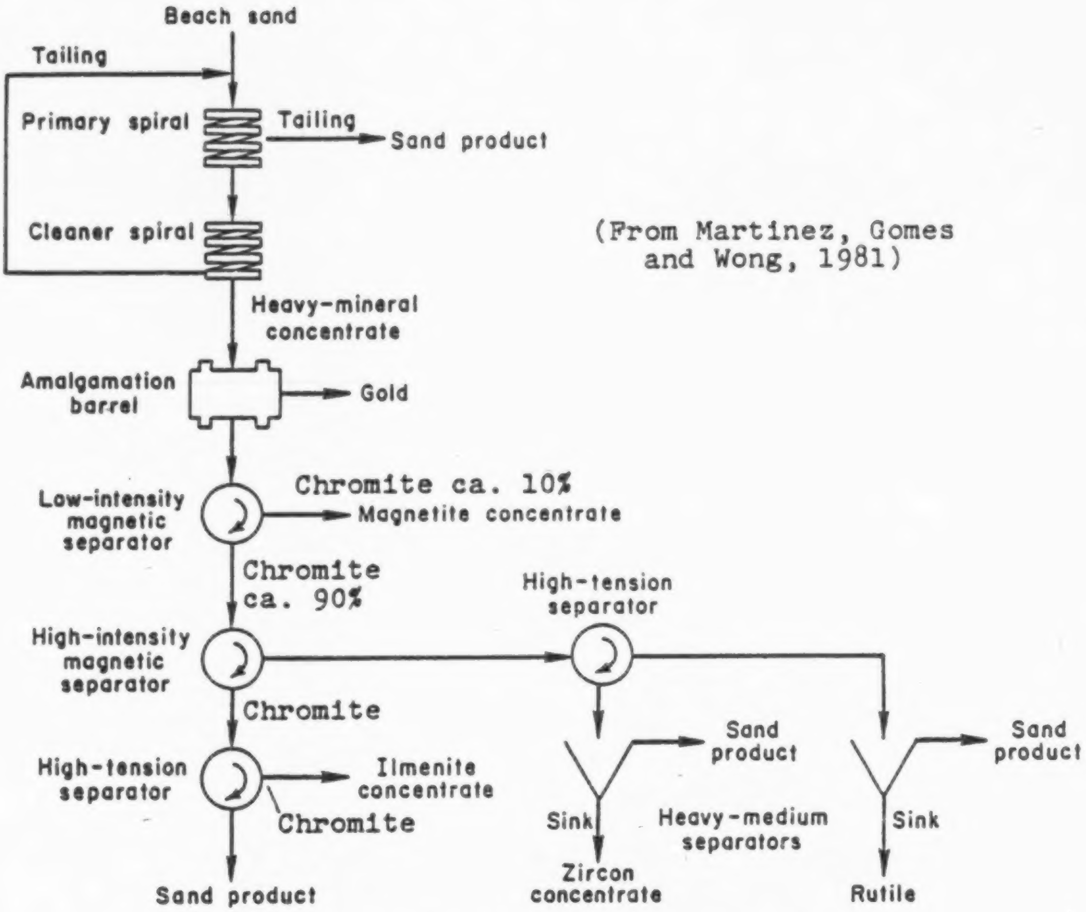

Fines from Lump Ore
Screen
2 x 1/4"
1/4" x 35M
- 35M
Heavy Media Separation
Jigs
Sorting Classifier
Slimes
Shaking Tables Treating Separate Sorted Fractions
Slime Table
T
C
T
C
T
C
T
C
Concentrate
Tailings

Fig. 4. Generalized flowsheet for the treatment of fines from lump chromite ores.

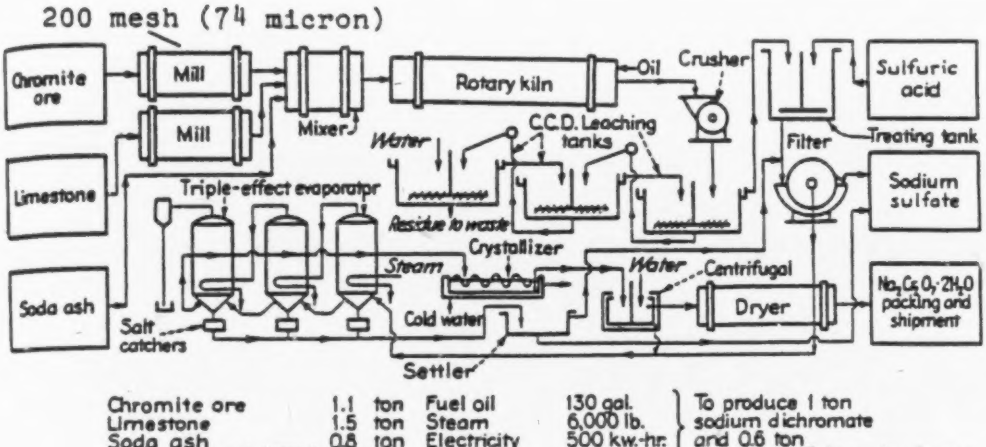
(From Aplan, 1985)



Beach sand
Tailing
Primary spiral
Tailing
Sand product
Cleaner spiral
Heavy-mineral concentrate
Amalgamation barrel
Gold
Low-intensity magnetic separator
Chromite ca. 10%
Magnetite concentrate
Chromite ca. 90%
High-tension separator
High-intensity magnetic separator
Chromite
High-tension separator
Ilmenite concentrate
Chromite
Sand product
Sand product
Sink
Heavy-medium separators
Zircon concentrate
Sand product
Sink
Rutile

(From Martinez, Gomes and Wong, 1981)

FIGURE 5. Sequence for processing Moclips Beach sand.



200 mesh (74 micron)
Chromite ore
Mill
Mill
Limestone
Mixer
Rotary kiln
Oil
Crusher
Sulfuric acid
Treating tank
Water
C.C.D. Leaching tanks
Filter
Sodium sulfate
Triple-effect evaporator
Residue to waste
Soda ash
Steam
Crystallizer
Water
Centrifugal
Dryer
$Na_2Cr_2O_7 \cdot 2H_2O$ packing and shipment
Cold water
Salt catchers
Settler

Chromite ore	1.1 ton	Fuel oil	130 gal.	To produce 1 ton
Limestone	1.5 ton	Steam	6,000 lb.	sodium dichromate
Soda ash	0.8 ton	Electricity	500 kw.-hr.	and 0.6 ton
Sulfuric acid, 66°Bé.	0.45 ton	Direct labor	15 man-hr.	sodium sulfate anhydrous

Fig. 6. Flowsheet for manufacture of sodium dichromate from chromite ore (Shreve and Brink, 1977).

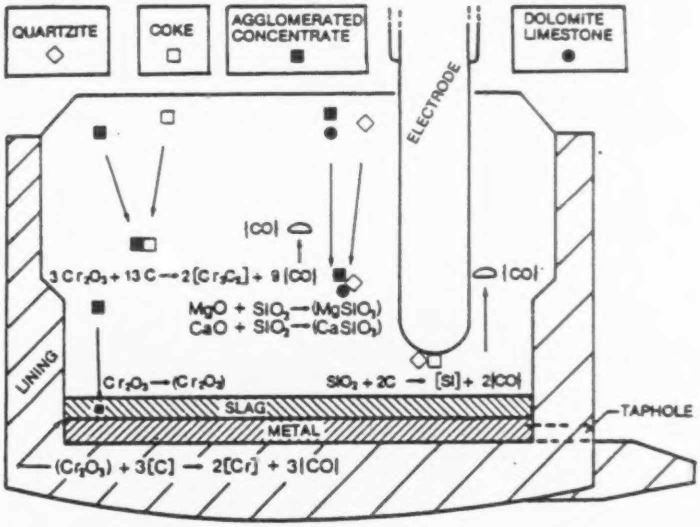

QUARTZITE ◇
COKE □
AGGLOMERATED CONCENTRATE ■
DOLOMITE LIMESTONE ●
ELECTRODE
|CO|
$3Cr_2O_3 + 13C \rightarrow 2[Cr_3C_2] + 9|CO|$
|CO|
$MgO + SiO_2 \rightarrow (MgSiO_3)$
$CaO + SiO_2 \rightarrow (CaSiO_3)$
LINING
$Cr_2O_3 \rightarrow (Cr_2O_3)$
$SiO_2 + 2C \rightarrow [Si] + 2|CO|$
SLAG
METAL
TAPHOLE
$(Cr_2O_3) + 3[C] \rightarrow 2[Cr] + 3|CO|$

FIGURE7 . Schematic representation of a submerged-arc electrofurnace process for reduction of chromite to recover high-carbon ferrochromium according to this study.

(From Dresler, 1986)

4. Polymetallic Sulfides and Oxides of the U.S. EEZ: Metallurgical extraction and related aspects of possible Future Development

Consulting and R & D in Extractive Metallurgy, Energy and Environmental Technologies

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POLYMETALLIC SULFIDES AND OXIDES OF THE U.S. EEZ: METALLURGICAL EXTRACTION AND RELATED ASPECTS OF POSSIBLE FUTURE DEVELOPMENT

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Appendix

January 1987

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POLYMETALLIC SULFIDES AND OXIDES OF THE U.S. EEZ: METALLURGICAL EXTRACTION AND RELATED ASPECTS OF POSSIBLE FUTURE DEVELOPMENT

Foreward

This is the fourth in a series of brief evaluations of metallurgical processing and related aspects of major mineral types found in the U.S. Exclusive Economic Zone (EEZ). The preceding chapters dealt with titanium heavy mineral (H.M.) and chromite H.M. placers and with phosphorite deposits. The occurrences of major interest are located in relatively shallow waters (i.e., in comparison with the polymetallic sulfides and oxides of the present chapter). Development of appropriate mining systems is thought not to involve unreasonable extrapolations from sea-going dredge technologies employed, mainly elsewhere in the world, for offshore sand and gravel and for tin and precious metal placers. The evidence is at least fair that mineral processing of the placer minerals and phosphorite can be handled by known and practiced methods, suitably modified and extended, although carrying out a normally preferred preliminary concentration at sea is clearly more problematic.

Activities directed toward polymetallic sulfides and oxides (in the latter category, particularly the cobalt-rich ferromanganese oxide crusts) are proceeding apace, so that it is difficult for an evaluation to be current with respect to the results of the latest cruises, laboratory tests, agency actions, etc. Accordingly, it is hoped that participants in the forthcoming Workshop will come prepared to summarize the latest major findings and developments in their respective areas of interest, for possible inclusion or mention in appropriate sections of the OTA project reports. In addition, comments and clarifications will be much in order for the questions raised and the perceptions formed in this necessarily cursory update and evaluation.

I. Recent Literature

The subject of marine hydrothermal deposits was comprehensively reviewed and their ore resource potential preliminarily evaluated by the Congressional Research Service in 1982 (Van der Voort and Mielke). Salient features and factors were sufficiently well recognized at that time, that subsequent work has changed the picture mainly quantitatively -- at least until quite recently. The CRS study responded to congressional interest in achieving the full scope of the Deep Seabed Hard Minerals Act of 1980, in whose evolution, interest in manganese nodules had figured prominently. Its authors cited cost figures compiled by Cruikshank (1981) for a manganese nodules venture, as possibly suggestive of the investment and operating costs that would be entailed in commercial development of subsea polymetallic sulfide (PMS) deposits.

A number of more recent reviews, theme issues of topical journals, compilations, and miscellaneous articles and announcements cover the published information on subsea polymetallic sulfides and oxides through mid to late 1985. Special mention is made of the updated comprehensive review of subsea mineral resources by McKelvey (1986). The Marine Technology Society Journal, fourth quarter, 1985 issue is devoted to marine minerals, as is Volume 5, Number 2 of Marine Mining (1985) and the forthcoming Volume 6, Number 2 of Marine Mining (1986). In addition, topical sessions in the published proceedings of Oceans '83 (Marine Technology Society and IEEE) contain a number of papers on marine polymetallic sulfides and oxides. To a diminished extent, this is true also of the proceedings of Oceans '84 and '85. The number of presentations devoted to this topic at the annual Offshore Technology Conference has similarly declined in recent years. Individual papers in the above general references contain, cumulatively, a great many references to the wider literature.

The proceedings of the Second International Seminar on Offshore Mineral Resources (organized by Germinal, Brest, March 1984) is replete with pertinent articles. Although not as recent, the 75th

Anniversary Volume of Economic Geology contains several outstanding papers in the subject area of this chapter. The draft reports of the NATO Working Groups on Marine Sulfides and on Manganese Nodules and Crusts were also consulted.

Several papers presented at last year's 16th Underwater Mining Institute, in Halifax, are germane to this and preceding chapters of the present assessment and emphasize Canadian developments and viewpoints. The papers are to be published in Volume 6, Number 2 of the journal Marine Mining. This year's UMI, in Biloxi, has scheduled sessions on marine mining technology and techniques and on marine mineral resources.

II. Polymetallic Sulfides

A. Subsea Occurrences

Draft compilations of discovered showings and deposits for marine polymetallic sulfides have been prepared, for occurrences offshore of British Columbia, by Samson (Canadian Oil and Gas Lands Administration, October 1985) and, worldwide, by Hoagland (Woods Hole Oceanographic Institution, May 1986). These compilations each contain numerous references to the original literature. A worldwide inventory of metalliferous sediments and polymetallic sulfides will be issued by the Geological Survey of Canada (Cross, 1987). According to its author, the compilation provides descriptions of the nature of the deposits in correlation with their geological settings.

For the Gorda Ridge area, offshore Oregon and northern California, additional specific and background information is supplied by the draft Federal EIS for a proposed polymetallic sulfide minerals lease offering (J.P. Zippin, Minerals Management Service, December 1983). The latest resource technical information should be forthcoming at the December 13, 1986 meeting of the Gorda Ridge Technical Task Force, in San Francisco. This and other activities reflect a high level of scientific interest in marine polymetallic sulfide deposits.

Published information on the subject consistently points to a direct connection between polymetallic sulfide (PMS) occurrence and hydrothermal activity associated with submarine volcanism. Sites for marine ore genesis of this general nature include hot spots and zones of seafloor spreading. In the latter category, three oceanic rift systems occur in the latitudes of Vancouver Island, Washington State, Oregon and northern California.

The three named ridge lines are sequential but displaced from one another, their extremities being connected by fracture zones. The Explorer Ridge lies within the Canadian EEZ; the Juan de Fuca Ridge falls half within the Canadian EEZ and half outside, beyond the limits (200 nautical miles) of the U.S. EEZ; and the Gorda Ridge is contained within the U.S. EEZ. Until very recently, no major PMS occurrences had been reported for the Gorda Ridge area, and it is hoped that new information on this topic will be forthcoming at the November Workshop. The decision to undertake prelease actions for the Gorda Ridge area, in advance of locating promising PMS occurrences there, implies a good deal of confidence in the association of PMS deposits with submarine volcanism and hydrothermal activity.

Conditions exist for the formation of PMS in other areas of the Pacific and in other oceans and seas, and occurrences have been found in a number of these locations. If divergent plate boundaries constitute the most likely sites for formation of major PMS deposits, then it appears that the great majority of such deposits will be found outside of the U.S. EEZ, including the EEZ's for the State of Hawaii and for island possessions and territories. One supposes that PMS deposits occurring outside of national EEZ's would be subject to exploitation on much the same basis as the ferromanganese oxide nodules of the deep ocean. Within the scope of the present OTA assessment of the mineral resource potential of the U.S. EEZ, however, the findings of exploration efforts in the Gorda Ridge area will have particular significance for the recognition of a national offshore PMS mineral resource.

In reviewing published information on findings of PMS occurrences to date, one gains the impression that the more spectacular manifestations of seabed hydrothermal ore genesis, such as the chimneys and open vents, may have been diverting attention away from locations and mechanisms possibly giving rise to deposits of more substantial extent and nearer-term exploitability. This impression is reinforced by the recent Canadian finding of a sediment-hosted PMS deposit in the Middle Valley section of the northern Juan de Fuca Ridge (Pasho, personal communication, 1986). The location, a circular area of high heat flow and mounded structure, was a previously detected Seabeam feature (Samson, 1985). A recent core of a few meters in length revealed typical zinc sulfide and copper sulfide mineralization.

A surficial sediment-hosted PMS deposit can be seen to have improved or nearer-term potential by comparison with consolidated massive sulfide deposits having the same composition with respect to recoverable minerals. According to Pasho (1986), because of the anticipated difficulty of mining and the lack of developed technology for the purpose, the massive-type deposits would likely require a high gold content to be of commercial interest, whereas the highest value element significantly present in subsea PMS samples has usually been silver. For PMS occurrences in unconsolidated sediments, there is the possibility of using or building upon technologies for mining and at-sea beneficiation that have been developed for the Red Sea muds.

By analogy with massive sulfide deposits of similar composition and possibly similar origin that are mined on land, and from reported observations and other information in the literature on the genesis of subsea polymetallic sulfide minerals, it is almost assured that their deposits will be found, on average, to be mineralogically and structurally complex. One can anticipate heterogeneity on both micro and large scale, with little uniformity among major deposits. One can also predict that PMS occurrences comprising or associated with spectacular features such as chimneys and other vents, and white and black smokes, will prove to be only small fraction of total subsea polymetallic sulfide mineralization. Inventories based on those surficial occurrences will be misleadingly low if, as suspected, the mineral zones lie predominantly at depth. Moreover, buried and overlain PMS deposits may be found at considerable distances from the active areas.

It remains to be determined by further exploration, including deep drilling, what is the significance in terms of tonnages of PMS deposits in sediments relative to consolidated, presumably largely buried, PMS deposits. There is probably no disagreement that access to and ultimate mining of surficial sediment-hosted deposits will be vastly easier. Certainly, underground mining far offshore would present an enormous engineering and economic challenge.

B. Terrestrial Deposits

1. Key Word Associations

The key to recognition on the part of extractive metallurgists of the common origin and, by extrapolation, similar nature of subsea PMS deposits and a major class of base metal sulfide orebodies found on land is the word 'complex'. This adjective was used above in connection with the mineralogy and structural features of subsea PMS deposits. The descriptor 'polymetallic sulfides' as used by marine geologists could be interpreted to mean the same as 'multimetal sulfides' which, in a strict sense, could apply to almost all sulfide ores. For example, the well-known porphyry copper deposits typically contain chalcopyrite, $CuFeS_2$, and pyrite, FeS_2, as the major primary sulfide minerals (to the extent of a few percent, disseminated in nonsulfide gangue minerals).

'Complex sulfides' has come to denote deposits whose sulfide mineral associations are more complicated than for simpler sulfide ores and whose metallurgical treatment is, consequently, difficult. The correlation between subsea polymetallic sulfide occurrences and terrestrial complex sulfide ores has important consequences for the types of beneficiation and extractive metallurgical processes that will be appropriate for the subsea deposits. The structure and marketing arrangements of that part of the metal-mining industry based on complex sulfides may also serve as guidelines for the future development of an offshore PMS industry.

2. Nature and Genesis

Complex sulfide deposits may be said to constitute a major class of mixed sulfide deposits, whose mineralogy is so intricate that "clean" separation, with high yield, into individual sulfide concentrates by normal beneficiation techniques is virtually precluded. The mineralogical textures of complex sulfides include various metallurgically adverse features as: interlocking of grains of different sulfide minerals on a very fine scale, mineral intergrowths, solid solutions for certain combinations of sulfides, and fine occlusions, as from exsolution. Many complex sulfide deposits are also "massive", in the sense of consisting predominantly of sulfide minerals rather than of nonsulfide gangue minerals such as silicates and carbonates. Often, the major gangue mineral in massive complex sulfides is the ubiquitous pyrite.

Complex sulfide deposits have been found in many parts of the world and in many geological periods. A pictorial representation from 1978 (as reproduced by Cases, 1980) of the volcanosedimentary genesis of complex sulfide ores shown in Figure B1 looks very much like more recent depictions by marine geologists of the genesis of subsea polymetallic sulfides. Similarity of origin is also evident in the following excerpt, which may be read with reference to Figure B1, concerning the geology of an important zinc-lead-copper-silver orebody near Bathurst in northern New Brunswick (E&MJ, 1981):

> "The massive sulfides are believed to be chemically precipitated sediments, and the irregular pyrrhotite-pyrite mass that cuts through the rocks underlying the massive sulfides are believed to represent a vent through which the ore elements ascended to the surface. Chemical precipitation is believed to have occurred above the footwall metasediments in an irregular shallow basin. The feeder zone is seen in underground diamond drill holes and workings at the mine. Stockworks of sulfide mineralization in rocks underlying the sulfide bodies are seen at both mines."

Metamorphosis and structural deformation add to the variability exhibited by terrestrial complex sulfide orebodies and anticipated for subsea PMS deposits. Covering with gossan (iron oxides resulting from submarine oxidation of sulfide minerals) or its equivalent may delay the recognition of associated sulfide orebodies. This was the case for the Bathurst area, in which occurrences of sulfides were overlooked for half a century (Luff, 1986). During this period, an iron formation was mined at intervals. Subsequently, more than 30 massive sulfide deposits have been identified in the region, of which two have been economically minable up to now: one an open pit mine and the other (Note), an underground mine.

Variability appears to be as much an attribute of complex sulfide ores as is complexity. Nevertheless, they most often contain some combination of economic values of copper, zinc and lead as their sulfides, with valuable silver and, sometimes, gold contents together with dominant iron sulfide. This usually means that the major sulfide minerals are chalcopyrite ($CuFeS_2$), sphalerite (ZnS), galena (PbS) and pyrite (FeS_2) or pyrrhotite ($Fe_{1-x}S$). Compositions typically fall in the ranges: 0.1-3% Cu, 0.3-3% Pb, 0.2-10% Zn, 20-50% S, 30-100 g/t Ag, and 0-10 g/t Au (Barbery et al., 1980). In some cases, total sulfide content of massive-type ore may exceed 90%.

Major sediment-hosted stratiform deposits with different host rock lithology are known (Gustafson and Williams, 1981) and mined, in addition to volcanic-associated massive sulfide deposits. Franklin et al.(1981) surveyed economic complex sulfide deposits of both major types and published the compositional diagram shown in Figure B2. They suggest that the apparent gap between copper-rich cluster and zinc-copper cluster is probably "an artifact of mining economics". This is notably the case for the massive sulfide deposits of Cyprus, whose predominant sulfide minerals are pyrite and chalcopyrite, but with significant zinc that is not recovered. There are also well-known ores of which the main or only economic mineral is pyrite (for its sulfur content). It will be interesting to learn, with further exploration, whether subsea polymetallic sulfide deposits generally conform to the compositional fields for terrestrial complex sulfide ores.

3. Economic Importance

Although complex sulfide deposits have been worked since ancient times, the copper mines of Cyprus being perhaps the most cited example, their numbers and economic importance have surged in recent decades. Steady decline in the average grades of large sulfide orebodies led to renewed emphasis in exploration for complex sulfides. Simultaneously, improved prospecting techniques led to the discovery of many complex sulfide deposits, which are often hidden. It should be noted that the majority of those found cannot presently be economically worked, although metallurgical developments have been intensive and wide ranging.

The relative economic importance of complex sulfide deposits can be judged from information in Table B1, as compiled in 1972 and reviewed by Cases in 1980. The figures show that, at the time, about 15% of copper production worldwide was from volcanosedimentary massive sulfide ores, reserves of which included about 7% of total copper reserves. The corresponding percentages are higher for zinc production and reserves.

Adverse base-metal markets in more recent years may have slowed the trend toward greater relative utilization of complex ores but has not reversed it. The situation here has probably been similar to that experienced by the primary nickel industry, with the rate of increase of dependence on oxide, mainly laterite, as opposed to sulfide reserves having been slowed or even temporarily reversed.

As illustration of the trend, consider one consequence to Japan of that country's increasing dependence on nonferrous metal concentrates from overseas, to feed its smelters and refineries. That movement has been given impetus by the dramatic appreciation of the Japanese yen over the past year, resulting in reduced production and some shutting down of nonferrous metal mining operations in Japan. What has been observed (Mohri, 1986), concurrently, is that imported copper concentrates now include a greater proportion of complex sulfide concentrates, containing a higher content of precious metals and of deleterious impurities such as arsenic, antimony and bismuth. As one consequence, Japanese smelters are having to develop complex sulfide metallurgy.

C. Processing Technologies

1. Topic Approach

The order of consideration will be from the most recently sampled and metallurgically tested type of PMS deposit, occurring in association with ocean rift systems, to the widespread complex sulfide deposits found on land. The intermediate case is that of metalliferous sediments of the Atlantis II Deep, the so-called Red Sea muds. It is too soon to speculate on the order of average mineralogical and metallurgical complexity of these PMS occurrences, because only a miniscule fraction of subsea deposits have been sampled, whereas terrestrial deposits exhibit a very wide range of complexity.

A review of copper, lead and zinc processing facilities and technologies was recently conducted by the U.S. Bureau of Mines (Norberg, 1985) as technical assistance to the Joint Federal/State Gorda Ridge Technical Task Force. That effort arose out of the December 1984 Task Force Meeting in Newport, Oregon. The report provides a useful summary of smelting, refining and leaching operations of this important group of nonferrous metals and includes inventories of producers, smelters and refineries.

The circumstance that unprofitability experienced by domestic copper, zinc and lead producers has led to reductions in capacity, some of a permanent nature, is too well known to require more than mention here. Domestic self-reliance in these metals presently decreases in the order Pb,Cu,Zn and, in the main, is a function more of limitations of base metal prices than of indigenous mineral resources. The U.S. share of world production of these and other major metals is expected to decline to the turn of the century. This general topic has been the object of much discussion and scrutiny, as is made clear in the substantive, as well as substantial, very recent Congressional Research Service Report (1986).

The BuMines review referred to above was initiated, and may have been completed, before PMS deposits were found within the Gorda Ridge, attesting to the general confidence in the validity of extrapolation of findings in other rift systems. The consequences of a possibly complicated nature of the PMS mineralogy, as an average for occurrences throughout the Gorda Ridge system, and consequently difficult metallurgy were not remarked. Possibly, this was the result of amenability, in laboratory studies, of a few samples of massive sulfide deposits from the Juan de Fuca Ridge to conventional mineral processing and extractive metallurgy.

2. Subsea Massive PMS Samples

Information on further samples and metallurgical tests of subsea polymetallic sulfides may be available before long. For the present, prior to the November 20 Workshop, results of metallurgical extraction tests are available for two ca. 150-g specimens from dredge samples recovered from the southern Juan de Fuca Ridge in 1982 (References 1 and 7 in Sawyer et al., 1983). The test specimens, in the form of slabs, were massive sulfide, in the sense of being comprised predominantly of sulfide mineralization. Zinc sulfide was the major mineral, to the extent of 88% and 55%, respectively, calculated as ZnS. Iron was higher in the former sample, 3% and 15% Fe, respectively, and copper and lead were minor in both samples. The copper and lead also occurred as sulfides so that, when ground, the materials were similar in form and overall chemical composition to ordinary zinc sulfide concentrates.

Sawyer et al.(1983) considered several nonsmelting alternatives for extractive metallurgical treatment of the samples, including conventional roast-leach-electrowinning; the Sherritt-Cominco process of pressure leaching with sulfuric acid and oxygen, as practiced at Trail, B.C.; a patented cupric chloride/hydrochloric acid leach; and a BuMines-investigated strong sulfuric acid leach process. The most promising approach was deemed to be chlorine-oxygen pressure leaching, as outlined in Figure C1 from a later BuMines report (Smyres and Carnahan, 1985).

On subjecting the two samples of massive PMS to the pressure leaching and cementation steps, zinc extraction of 99% in leaching and silver recovery of 97% in the zinc cementation product were obtained. These results are consistent with the chemistry of the process: the leach conditions are such as to dissolve, at least transiently, practically all constituents of the sulfide "concentrate" except for silica; and elemental zinc can precipitate from solution those electrochemically less active metals, including silver, copper and many others.

Major features of the chlorine-oxygen leach process include control of chloride supply so as to promote hydrolysis and precipitation of ferric ion into the leach residue, or tailings, rather than solubilization as ferric chloride, and regeneration of chlorine as the side product of the fused salt electrowinning of zinc chloride. There are also potential drawbacks to the approach. Cementation with metallic zinc is commonly and effectively used for zinc electrolyte purification in conventional zinc electrowinning from sulfuric acid solution. It is an expensive step in terms of use of recycled zinc, which subtracts from production, and in terms of the separation and recovery requirements for the value metals in the cementation product. In addition, while the d.c. electrical energy consumption might be lower for zinc electrowinning from fused chloride, as compared with aqueous sulfate, the total energy requirement might not be less. This is because of the heat requirement for water evaporation in $ZnCl_2$ crystallization and drying and the heat losses attending cell operation at fused-salt temperatures.

If the subsea PMS deposits turn out, on average, to have a more complex mineralogical makeup than the two massive sulfide specimens discussed above, with intricate associations of the sulfide minerals and higher amounts of nonferrous values in addition to zinc, the sulfide dissolution treatment tested by Sawyer et al.(1983) could have serious problems. In particular, costs of cementation by zinc and subsequent separation and recovery of larger quantities of copper and lead, for example, could become prohibitive. Treatment and disposal of the ferruginous, sludgy residue from chlorine-oxygen pressure leaching would not be a trivial undertaking, whether for disposal in a secure landfill or disposal at sea.

There might be some advantage in considering a more selective leach treatment. An example of an approach is a nonoxidizing hydrochloric acid pressure leach (Figure C2), by which pyrite is substantially unattacked, chalcopyrite is reconstituted into simple copper sulfides, and other major nonferrous sulfides, including zinc sulfides, are dissolved as their chlorides (Harvey and Dudas, 1977). Even modified to emphasize recovery of zinc as the principal product, the Figure C2 process also has major deficiencies, including materials for containment of heated, strongly acidic chloride solution under nonoxidizing conditions and the requirement for regenerating HCl from the leach solution. Possibly, some elements of the flowsheet could be adapted to the initial separation of a bulk sulfide concentrate.

Carnahan (1985), in a recent paper, concluded that sufficient data had been collected on metalliferous sulfides from the Pacific Ocean and Red Sea to permit preliminary assessment of potential processing techniques. In that paper, it was noted that the bulk of the material in marine sulfide deposits is of too low grade for direct smelting, i.e., without prior beneficiation. It was further indicated that the Pacific Ocean deposits, while containing large amounts of pyrite and other worthless sulfides, have grain sizes in the range suitable for froth flotation.

The general treatment proposed by Carnahan (1985) for PMS deposits of the Pacific Ocean consists of

blending to a target composition
crushing and grinding to liberation size
bulk flotation, with disposal of iron sulfide tailings
selective flotation of copper and zinc sulfides
smelting to metal of the separate concentrates.

This is an essentially conventional mineral processing and extractive metallurgy (MP&EM) procedure for simple copper-zinc ores. Carnahan also mentions the roast-leach-electrowinning alternative for low-lead zinc concentrates and the zinc-lead blast furnace for concentrates containing both sulfides.

By contrast, the Red Sea muds are indicated by Carnahan to be too fine-grained for efficient beneficiation by physical methods to smelter-grade concentrate. For that material, he recommends a chloride-based hyrometallurgical process, such as the chlorine-oxygen leach described earlier (Sawyer et al., 1983) in connection with processing the massive sulfide specimens from a Juan de Fuca deposit.

The extrapolation from very limited resource and MP&EM data on subsea massive sulfides to a conclusion in support of largely conventional processing may not be sustainable. Specific inclusion of an ore blending step is perceptive in view of potentially large physical, chemical and mineralogical variability of the resource. Bulk, i.e., nonselective, sulfide flotation preceded by comminution, as required, should be feasible in the general case, but feasibility of selective flotation to produce smelter grade concentrates cannot be assumed with any confidence. Uncertainty with respect to this step and subsequent smelting to metal will become apparent in considering metallurgy of Red Sea muds and terrestrial analogs of subsea massive polymetallic sulfides.

Carnahan (1985) made the interesting observation that the subsea massive PMS specimens were porous, owing to the action of worms and other biota. This observation and direct observations of bacterial activity in-situ suggest a possible application of bioleaching in the beneficiation process (cf. Carta et al., 1980). Bioleaching has been studied and applied to a variety of ores and concentrates in a number of MP&EM contexts and is generally characterized by slow kinetics. Perhaps if microorganisms adapted to higher solution temperatures were available, that aspect could be improved sufficiently to make "biohydrothermal" treatment a possibility. One suspects that, taking account of the hydrothermal and bioactive conditions which play a role in the formation of subsea PMS deposits, controlled modification of those conditions might be a key to unlocking their mineral values. Variability of formation conditions of marine polymetallic sulfides in time and space presents a complication to this as well as to more conventional conventional metallurgy.

It may not be productive to speculate further on processing technologies until more has been learned about the numbers, types, sizes and geological settings of subsea PMS deposits. It is to be hoped that funding will be available for this important work, including not only reconnaisance and surficial sampling, but also drilling for determination of vertical variability and extent. The thoroughness of exploration and delineation of the Red Sea sedimentary deposits may be an appropriate model for further similar activities directed toward subsea PMS deposits of the U.S. EEZ.

3. Red Sea Metalliferous Muds

Evolution of the long-term program of delineation of the resource potential of metalliferous sediments of the Red Sea deeps and the results of that program have been well described in the literature. Evolution of a plan for exploitation of the Atlantis II Deep sulfide muds and development of workable mining and processing methods have also been described. This information will be of particular interest for assessment of the commercial potential of sediment-hosted subsea PMS deposits, such as that recently identified in the Canadian EEZ and, presumably, others waiting to be discovered.

The attempt will not be made here to summarize the, by now, large amount of published information on the Red Sea muds of primary commercial interest. Mustafa et al.(1984) provide a comprehensive update of the Saudi-Sudanese project and McKelvey (1986), in his latest survey, summarizes chemical analyses and mineralogy of sulfide facies of portions of the Atlantis II sediments. Average metalliferous sulfide compositions differ widely from deep to deep. The sediments in the Atlantis II Deep, in particular, are highly stratified both in depth and laterally, in a setting of complex seafloor morphology. The oxidic layers may contain as much as 60% iron or 30% manganese, and the sulfide layers contain, in addition to pyrite, major zinc and associated copper and silver. As of 1980, the content of economically recoverable gold had not been verified (Fletcher and Mustafa, 1980), but recent plans call for production of gold and possibly other trace elements including cobalt, cadmium and molybdenum (Mustafa et al., 1984). Lead seems to be low, of the order of 0.1-0.2%, and it will be interesting as well as significant to learn whether other subsea PMS sulfides are also lead-poor. Proven metal reserves are said to be sufficient to support, at approximately 50% recovery, 60,000 t/yr zinc production, plus associated value metals, for 16 years. This figure is consistent with a mining rate equivalent to 200+ t/day zinc metal, with a 70-75% stream factor.

Major-element and silver analyses of Atlantis II sediments are given in Table C1. The multidimensional variability in composition (with depth as well as with surface position as shown) points to the desirability of a selective mining capability, by means of a sensor-equipped dredge head (Amann et al., 1973). Without that as yet to be developed capability, an average grade mud consisting of 3-6% zinc, up to 1% copper and about 50 ppm silver will be mined. Today, the composite mined sediments would not be considered of unusually high grade as a zinc ore or as a copper ore, but the combined values make for a more interesting economic target.

The fine state of subdivision of the Red Sea metalliferous sediments is a major factor in their mining, handling and processing. Note, in Table C2, that only a few percent of the particles are larger than 10 µm; it is only for this "coarse" fraction that mineral identifications have been made. Pyrite, sphalerite and chalcopyrite, with considerable intergrowth, have been identified in the sulfide fraction and iron hydroxide, mixed iron-manganese carbonate and other carbonates, anhydrite (calcium sulfate) and silicates, in the nonsulfide fraction (Fletcher and Mustafa, 1980). Major nontronite (an iron-rich aluminosilicate clay mineral) has also been identified. It has been reasonably well established that the valuable zinc, copper and silver, as well as part of the iron, occur as sulfides.

The deeper and denser muds have been described as having the consistency of boot polish, considerable dilution by seawater jetting being required to reduce slurry viscosity sufficiently for pumping to the surface. The corresponding volumes of slurry to be handled in the production of approximately 250 t/day zinc plus copper are very large indeed (Figure C3). However, they do not exceed the aqueous flow in the largest solvent-extraction copper-electrowinning plants.

Development of an effective mining system for the Red Sea muds has been an impressive achievement. Out-of-the-ordinary difficulties encountered include the depth, in excess of 2000 m; the overlying hot, concentrated brine layer; the irregular topography of the bottom; and the complex stratigraphy and unfavorable rheological properties of the sediments. In addition, the metalliferous sediments of interest for mining were found to be partly consolidated, with hard anhydrite lenses at various depths and lateral locations (Mustafa et al., 1984). This last difficulty was met by a vibratory screen suction head which, together with seawater jetting, disintegrates the consolidated muds and induces flow and dilution of the thixotropic, brine-impregnated sediments. It is noteworthy that the airlift design of mining system was considered and later rejected in favor of pumping (one submerged pump each for pumping of dilution seawater to the bottom and for pumping mud slurry to the mine ship). Energy requirements are said to be lower for pumping (Fletcher and Mustafa, 1980). Moreover, oxidation of the finely divided and reactive sulfide particles could be a problem in an airlift pipe. It will be recalled that unexpected occurrence of consolidated material, requiring elaboration of the mining head, was also encountered in the development of the onshore Santo Domingo phosphate placer deposit. It would seem prudent to anticipate similar mining difficulties, when estimating the development potential of new sediment-hosted deposits of the EEZ.

The fine particle size of the Red Sea muds that have been mined on a pilot basis would preclude normal flotation techniques as a means of preparing a bulk sulfide concentrate, i.e., separating the sulfide minerals collectively. Fortunately, dilution of the host brine with ordinary seawater appears to promote selective agglomeration of the highly reactive, fine sulfide particles on the addition of flotation collector (Fletcher and Mustafa, 1980). The scheme for carrying out the bulk flotation on board ship is shown in Figure C4.

Scaled up to commercial size from the pilot operation, the flotation would produce 700 t/day bulk sulfide concentrate from 10,000 t/day mud solids, both on a dry, salt-free basis. Note the comparability, in terms of tonnage of ore solids mined, with manganese nodules mining scenarios.

Problems in on-board flotation owing to ship's motion have apparently been satisfactorily solved, thanks in part to test operation of a flotation cell in conjunction with ship's motion simulation, at the Warren Spring Laboratory. Selective flotation to "clean" zinc and copper concentrates for conventional smelter feed is not feasible, however, because of the intimate sulfide mineral associations, as in intergrown ZnS and FeS, i.e., (Zn,Fe)S. Even the bulk flotation for preconcentration is not efficient in terms of overall recovery (ca. 70% according to Figure C3) and reagent consumption (Amann et al., 1973). Earlier considerations of pyrometallurgical treatments of the bulk concentrate, involving controlled roasting or smelting with the production of a. blister copper (impure copper metal) or copper matte (molten copper sulfide for conversion to blister) and b. zinc oxide for leaching/electrowinning or slag for fuming to zinc metal, were not carried through. Instead, attention turned to hydrometallurgical methods, especially sulfuric acid and chloride leaching. Although reactivity was adequate with both lixiviants, a chloride leach route was determined upon for various reasons, including excess sulfuric acid consumption.

Davy McKee developed a chloride hydrometallurgical process (Adamson and Naden, 1983; patent assigned to the Saudi-Sudanese Red Sea Joint Commission) comprising leaching the complex sulfide material at 100-180 °C with an acidic chloride solution containing magnesium chloride. Oxidative leach conditions result in formation of sulfur and rejection of iron in the form of oxide or hydrous oxide. The nonferrous metal values are recovered by liquid-liquid extraction, and magnesium chloride in the raffinate (solution remaining after solvent extraction) is converted by oxyhydrolysis (relatively high temperature reaction with air and steam) to magnesium oxide and hydrochloric acid for reuse in the process. This

Davy McKee single-stage oxidizing chloride leach process has some features in common with the BuMines leach process described above (Figure C1). Evidently, projected economics were not favorable, and other chloride leach routes were considered.

Recently (Mustafa et al., 1984), it was decided to go with the Comprex process of Técnicas Reunidas (Madrid). Técnicas Reunidas have operated an approximately 1 t/day demonstration plant for their pressure leach process since late 1978, including a campaign with Red Sea mud bulk concentrate supplied by Pressag-Red Sea Commission (Nogueira et al., 1980). Compositions of some typical bulk concentrates from complex ores that have been tested by the Comprex process are given in Table C3. It is noted that the Red Sea mud bulk concentrate does contain measurable lead and that gold is not shown as a recoverable value. In the same year (1980), however, Fletcher and Mustafa had indicated that a gold recovery step would be included in the metallurgical process finally chosen.

The main features of the Comprex process are illustrated in Figure C5. Major raw materials include oxygen and water for pressure leaching, crushed limestone for sulfuric acid neutralization, and ordinary salt for leach-extraction of silver and lead from the ferric oxide pressure leach residue. Temperature and oxidation intensity are sufficiently high in the pressure leach step to completely oxidize the metal sulfides, including pyrite, to sulfates and to oxidize and hydrolyze the iron to precipitate iron oxide as hematite, Fe_2O_3. The ferric oxide residue is more readily filterable and, also stabler on disposal, than hydrous ferric oxides usually obtained and more readily permits recovery of lead and silver. Copper and zinc are recovered by solvent extraction and electrowinning, the latter from a mixed sulfate and chloride solution by the Zincex process (Figure C5). The final liquid waste stream consists of a pH 8.5, 45-55 g/l NaCl solution, which could presumably be disposed of in the sea. Alternatively, the moderately saline effluent could be sent to a solar evaporation pond.

Results of a 1979 economic analysis of the Comprex process as applied to Spanish complex pyrite ores are summarized in Table C4.

The analysis was based on toll refining of the concentrates which, for the composition shown, were estimated to have a value of $US 252 per tonne. A very favorable pre-tax profit was forecast, and estimated cash flow was commensurate with a 2½-year payback. The presence of much less lead in the Red Sea mud bulk concentrate than in the base-case bulk concentrate from complex pyrite ore should not affect economic feasibility appreciably -- certainly not as much as the 50% lower copper content of the Red Sea mud bulk concentrate (Tables C3 and C4).

Let us assume that if Red Sea bulk concentrates are assigned values commensurate with those of commercial complex sulfide concentrates of comparable composition, their metal values can be won at a profit. The treatment process could be the established Comprex process or one of the other hydrometallurgical processes, such as a modified CLEAR process (Duval, U.S.A., now closed down), or the developmental Minemet process (France) or Elkem process (Norway), that have been considered for the Red Sea enterprise (Fletcher and Mustafa, 1980). Recent-year depressed base-metal and silver prices aside, which have undoubtedly decreased the value of the bulk concentrate in constant dollars, the economic feasibility of producing the identified reserves of Red Sea muds would seem to hinge on the total cost of producing the bulk concentrate relative to its price in the market. Red Sea Commission figures show a total venture net revenue of $US 2,240 million (Mustafa et al., 1984).

There are reasons to be skeptical of the profitability of a one-time venture of the nature described: for one, the limited proven reserves, with production life further shortened by low recovery in mining and in bulk flotation (perhaps 50% in the former and not more than 70% in the latter operation); for another, the undoubtedly considerable total cost of the extensive and prolonged development. Thirdly, the operation would be capital- and energy-intensive in all of its major phases, from mining to metal production. This is evident for the mining operation, and an economic analysis showed that energy costs for flotation and processing of the concentrates would amount to 50% of total operating costs (Mustafa and Amann, 1978). A mitigating factor is that most of the energy requirement at sea would be supplied by diesel generators and on shore by oil- or natural gas-fired power plants, and these fuels should be as cheap in the region as most anywhere on earth. Opposed to this is the disadvantage, for the same region, of having to provide supporting infrastructure.

There may be other factors in a future decision to follow through with an integrated process, besides the obvious one of being enabled to gauge profitability on the basis of overall balance sheet. Similar other factors may have played a role in the recent commissioning of a copper mining, smelting and refinery complex in Oman (Elliott and McCririck, 1985). There, natural gas from fields in the southern part of the sultanate is used for generation of plant electricity and steam, effectively constituting a subsidy. It is doubtful that copper is being produced at a competitive price, but part of the country's industrial expansion aims are being realized.

Possibly the best chance of realizing profitability on the part of a follow-up commercial enterprise based on the Red Sea development project to date would be extension to additional subsea sediment-hosted deposits. For a relatively small number of promising deposits, a single pioneer entity might do better by mining in different areas sequentially than would separate entities by mining in their respective EEZ's only. In the latter case, a pioneer Red Sea enterprise might ultimately turn a net profit through the sale of technology and know-how.

4. Terrestrial Complex Sulfides

Inclusion of this section reflects a strong hunch that, in the main, subsea polymetallic sulfide deposits will prove to be complex, as defined above. The implication here is that the specimens retrieved from the Juan de Fuca Ridge and subjected to metallurgical amenability testing in 1983 are likely to be unrepresentative. This is not to suggest that many more samples of PMS showings will not be similarly blocky, relatively coarse-grained massive sulfide. Rather, the average mineralogical make-up of minably large deposits could well be more complicated.

Because of the uncertainty with respect to composition and mineralogy, not to mention geological settings, of future PMS resources of the EEZ, a detailed account of MP&EM practices and problems for terrestrial complex sulfide ores would be premature. A related reason for restricting discussion to general aspects, illustrated by random examples of practice, is the enormous variability of complex sulfide orebodies. Effectively, there is no typical, detailed metallurgical sequence nor even a common strategy for their exploitation.

Over the past decade, roughly concurrently with the interest and excitement of discovery and exploration of hydrothermally active systems on the sea floor, the more prosaic complex sulfide orebodies on land have been subjected to concerted scrutiny. In addition to a number of individually published studies and reviews, there have been three major symposium volumes devoted to MP&EM of complex sulfide ores. These are "Complex Metallurgy '78" (IMM), "Complex Sulfide Ores" (IMM, 1980) and "Complex Sulfides" (TMS, 1985). The 1980 volume is especially recommended for historical notes and valuable insights.

Known complex sulfide deposits are numerous and tend to be relatively small. There are a few complex sulfide orebodies that are large enough and rich enough to support an integrated concentrator-smelter-refinery operation. Mount Isa Mines (Australia), for example, produce copper and lead metal and zinc concentrate from copper and silver-lead-zinc ores. For a number of complex sulfide orebodies, concentrates are treated at a custom smelter-refinery. The example shown, in Figure C6, is that of Boliden (Sweden). The least encountered arrangement is the use of hydrometallurgy to treat complex concentrates, as at Cominco's zinc-lead-copper operation at Trail, B.C.

It is not meant to imply that the Boliden plant (Figure C6), whose configuration and operation are the result of 50+ years of development (Barbery et al., 1980), is typical or representative of smelter-refinery complexes; the main purpose of showing the simplified flowsheet is to illustrate the variety of feeds, intermediates and products that may be encountered. These all must be dealt with in some way.

There are two additional major alternatives for dealing with a complex sulfide deposit, of which the first is to leave it in the ground, pending development of suitable technology and/or metal prices. This is the most frequently chosen of all alternatives. Where, as often, pyrite is the dominant sulfide, where production of the nonferrous metals would not pay for the mining and processing, and where cheaper sources of sulfur and sulfuric acid are not available, the complex sulfide may be treated as a sulfur + iron ore (Figure C7). This is done by roasting the "dirty" pyrite to produce sulfur and/or sulfur dioxide and pyrite cinders, an impure iron oxide. The most familiar current examples of the practice are in the pyrite belt of the Iberian Peninsular, with its numerous deposits of cupreous pyrites. Some clever extractive metallurgical processes have been devised to extract the nonferrous metal values from pyrite cinders and to produce an adequately pure, synthetic iron ore. The hydrometallurgical process of Duisberger Kupferhutte for treating pyrite cinders, recovering "everything from the pig except its squeal", was an impressive example of such recovery. Unfortunately, the commerical operation could not be sustained in the face of falling base-metal prices. It is reasonable to assume that treatment of subsea PMS deposits as pyrite ores, where applicable, would not be an attractive alternative.

Basically, assuming that the total contained value of the complex sulfide deposit is interesting, several treatment strategies need to be weighed. This is not a matter of engineering, economic and business analyses only, but requires a knowledge of the behavior of the material in various treatment sequences. That knowledge, in turn, is derived from delineation (three dimensions) of at least the operating reserves portion of the deposit and from metallurgical testing of individual and composite ore samples. Among the metallurgical strategies to consider, as part of the broader package (Figure C7), are whether to attempt production of a bulk concentrate (containing all of the nonferrous value-metal sulfides), or of selective concentrates, or of a semi-bulk concentrate plus a selective concentrate.

The concentration or beneficiation method of choice for complex sulfides is flotation. Application of column flotation technology for the purpose is being viewed with increased interest. Regardless of the physical techniques used in flotation, however, the intricate mineral associations that are usual in complex sulfides cause problems in flotation separation in one or more of several ways. The upshot is that selective (another term is 'differential') flotation usually entails low recoveries of value metals in the selective concentrates, which are usually 'dirty' (i.e., contain higher than optimum percentages of other minerals), rather than 'clean' as desired for smelting to metal. The problem is illustrated for Cu-Pb-Zn sulfides by the grades and recoveries tabulated in Table C5. Recoveries are generally much better in the production of bulk concentrates, but then one has effected only a partial separation (analogous to preconcentration for heavy-mineral sands).

Table C6 compares ranges of elemental percentages acceptable to smelters treating copper, lead, zinc, and mixed concentrates with those obtainable in selective concentrates from complex sulfide ores. In the table, 'ISP' refers to the Imperial Smelting Processes Ltd. zinc-lead blast furnace concentrates. The ISP and other pyrometallurgical processes for sulfide concentrates (among them, Outokumpu flash smelting, Siromelt technology, electric furnace smelting, and others) are being worked on to adapt them to more complex concentrates. In general, however, the penalties exacted by the smelter for not meeting specifications with respect to major metals, and also with respect to troublesome impurities, are significant. Some ranges of smelter payments and penalties are collected in the Appendix to this chapter. These penalties, together with low recoveries in flotation,have been sufficient reason to forestall development of many complex sulfide orebodies, even of relatively high grade.

A good deal of development has gone into combatting the inverse grade-recovery relationship for production of selective concentrates from complex, fine-grained sulfide ores. A major direction of effort consists of producing a bulk concentrate with high (90% or more) average recovery of all valuable metals and applying new extractive metallurgical technology to treat the resulting complex material. The economic comparison is between revenues to the mine based on a. estimated processing costs for treatment of bulk concentrate by the new technology and b. processing costs for the preparation of selective concentrates and for their processing in modern zinc, lead and copper smelters. Account is taken of the overall yields of the principal nonferrous metals of the ore, as well as of important by-products, especially silver and gold.

For illustration, we turn again to the Brunswick Mining and Smelting (BMS) complex massive sulfide orebody and concentrators (for concreteness, with no suggestion of typicality but with aspects of relevance to processing of subsea PMS deposits). Ore reserves at the still-producing No. 12 underground mine are estimated at 104 x 10^6 metric tons (t) grading 3.7% lead, 9.0% zinc, 0.32% copper, and 98 g/t (2.8 oz per s.t.) silver (Luff, 1986). At a mining rate of 3.5 x 10^6 t/yr, the Brunswick mine is one of Canada's largest silver producers. Silver mineralogy is very complex, with indication that all the major ore sulfides are silver carriers. As an aside, it may be noted that the present mining rate (about 10,000 t/day) is in the same range as that customarily considered for a manganese nodules mining venture (Cruikshank, 1981) and, by extension, for a polymetallic sulfides venture (Van der Voort and Mielke, 1982).

Mining and milling costs are privileged information, but an approximate total is $Can. 35 (Schnarr, personal communication, 1986), or about $US 25 per (metric) ton of concentrates. A rough breakdown is (in $US): mining, $\$14._4$/t; milling, $\$8._6$/t; and administration, etc., $\$2._2$/t. The indicated approximate mining cost is probably an average for the recently depleted No. 6 (open-pit) mine and the No. 12 deposit, mining costs presumably having been lower for the former.

BMS presently produce selective zinc, lead and copper concentrates and a bulk concentrate for processing in an ISP plant. The ore value (i.e., of the total content of valuable metals at their market prices) is about $116/t distributed as follows (Craigen and Schnarr, 1985): Zn 67.2%, Ag + Au 16.5%, Pb = Cu 16.3%. Actually, the gold content of BMS ore at about 0.2 g/t is too low to justify addition of a cyanide circuit, so that the indicated 16.5% percent of ore value probably reflects mainly silver.

Bulk concentrates from three New Brunswick ores, including the BMS ore, have much higher iron and much lower zinc contents, as well as significantly higher lead, copper and silver values, than the present zinc concentrate. Processing of the bulk concentrates by either of two high-recovery processes is estimated to result in a net payback. The processes are Sherrit Gordon's Pressure Sulfuric Acid (PSA) Leach process and the New Brunswick Research and Productivity Council's Sulfation Roast Leach (SRL) process.

The two sulfate processes have been tested on bulk concentrates prepared for the purpose from New Brunswick ores and are estimated to have the following capital and operating costs (Craigen and Schnarr, 1985), based on producing 100,000 t/yr zinc from BMS bulk concentrate:

	SRL	PSA
Capital cost (10^6 \$US)	181	147
Operating cost (10^6 \$US/yr)	34.6	34.7

The estimated investment and production costs appear to be favorable by comparison with corresponding costs for treatment of Red Sea bulk concentrates by the Comprex process (Table C4), when the latter estimates are adjusted to the same annual zinc production.

The differential in net revenue for bulk over selective concentrates increases with average metal prices, and that is the key to the adoption of new technology to replace existing smelting procedures for complex sulfides. A number of nonsmelting processes have been developed in addition to the above two sulfate routes and the two chloride routes mentioned earlier. A useful summary, with their status as of 1980, is presented in Tables C7 and C8.

5. Addendum

Informal current-activity reports at the November 20 OTA workshop by T.G. Carnahan (Reno) and B.W. Haynes (Avondale) of the U.S. Bureau of Mines included mention of PMS process studies. Carnahan amplified results of the laboratory work summarized in §C2 by observing that some marine sulfide deposits will not be amenable to beneficiation to smelter-grade concentrates. Hydrometallurgy would then be indicated, because of its ability to treat lower grade and mixed sulfides. In particular, Carnahan recommends oxygen pressure leaching and chlorine-oxygen leaching for zinc sulfide concentrates having a low chloride content and a high chloride content, respectively; ferric chloride - cupric chloride leaching would be appropriate for most copper concentrates from marine sulfide deposits. Haynes noted the existence of a study to determine the feasibility of processing larger quantities of zinc-rich PMS from the Juan de Fuca Ridge and anticipated that results of testing by three metallurgical routes would be available by mid-1987.

D. Tables and Figures, Section II

Table B1. Copper: economic importance of various types of deposit[1]

Type of deposit	*Total production, contained metal, $t \times 10^6$*	*Reserves, $t \times 10^6$*	*Average grade, %*
Porphyry copper	95	305	0.8
Cupriferous shales and sandstones	50	150	2.2
Massive sulphide deposits in volcanic sediments	29	37	1.35
Ni–Cu sulphide deposits	6.5	9	0.7
Miscellaneous	8.2	10.2	–
Total	188.7	511.3	

(From Cases, 1980)

Table C1. Composition of sediments from Atlantis II Deep

Area	Element, %				
	Zn	Cu	Ag	Fe	S
SW Basin	6.03	0.84	0.011	26.4	5.5
SE Passage	3.05	0.79	0.009	29.5	3.7
SW Passage	2.66	0.57	0.009	29.0	2.9
E Basin	2.65	0.57	0.007	28.6	3.6
W Basin	1.78	0.39	0.011	30.1	2.5
Central sill	1.74	0.54	0.009	25.3	3.1
N Passage	2.53	0.52	0.007	27.5	3.6
N Basin	0.89	0.21	0.005	21.8	2.1

(From Fletcher and Mustafa, 1980)

Table C2. Particle-size distribution of sediments

Size range, μm	%
+ 20	2
+ 10 - 20	3
+ 2 - 10	17
+ 1 - 2	26
+ 0.5 - 1	21
+ 0.125 - 0.5	11
- 0.125	20

(From Fletcher and Mustafa, 1980)

Table C3. Typical compositions (%) of bulk concentrates from different complex ores

	Cu	*Zn*	*Pb*	*Fe*	*Ag, g/t*
Sotiel	3	34	7	16	200
Aznalcollar	3	26	10	19	350
Rio Tinto	9	28	7	20	250
Aljustrel*	7	30	7	18	220
Red Sea mud†	3.5	32	0.6	16	450

* Material supplied by Comissão para o Lançamento do Programa do Aproveitamento Integrado das Pirites, Portugal.

†Raw material supplied by Preussag–Red Sea Commission.

(From Nogueira et al., 1980)

Table C4. Results of an economic feasibility study on the Comprex process applied to bulk flotation concentrates

Bulk flotation concentrate	*212 800 t per year*
Copper	7.05%
Zinc	30.90%
Lead	9.47%
Silver	308 g/t
Concentrate value	$U.S. 252/t (February, 1979)

Chemicals and utilities: consumption per tonne of bulk flotation concentrate

Sodium chloride	1800 kg	Process water	30 m^3
Limestone	1000 kg	Sea water	8 m^3
Sulphuric acid (100%)	86 kg	Cooling water	36 m^3
Hydrochloric acid (19° B)	24 kg	Fuel oil	55 kg
LIX 64N	80 g	Electric power	1500 kWh
Amberlite LA-2	150 g	Steam (export)	-1900 kg
Isodecanol	50 g		
D2EHPA	326 g		
Kerosene	10 kg		
Activated charcoal	323 g		

Results from economic study (February, 1979; total investment cost, $U.S. 126 000 000)

	$U.S./t concentrate	*$U.S. × 10^3 per year*
Raw material cost	252	53 626
Production cost	174	36 956
Total cost	426	90 582
Revenue	649	138 108
Profits (before tax)	223	47 526
Cash flow	263	55 949
Payback time 2.26 years		

(From Nogueira et al., 1980)

Table C5. Bulk concentrate production compared with selective flotation

Aznalcollar (see Fig. 4 for details of selective flotation)	*Grade, %*			*Recovery, %*		
	Cu	*Pb*	*Zn*	*Cu*	*Pb*	*Zn*
Bulk concentrate	3–4	6	25	83	55	90
Example from Georgeaux *et al.*[8]						
Selective concentrates						
Cu	19	3	4	61	3	1.5
Pb	0.5	50	2	2	55	1
Zn	1	2.5	50	13	9	72
Bulk concentrate	1.8	5.8	19.2	80	70	90
Example from Stemerowicz and Leigh[8]						
Feed	0.24	3.95	8.73			
Bulk concentrate	0.7	10.75	30.0	77	87	95

(From Barbery et al., 1980)

Table C6. Average values acceptable to smelters and mean contents obtained from finely disseminated complex sulphides (From Cases, 1980)

	Concentrate	*Cu, %*	*Pb, %*	*Zn, %*
Values accepted by smelters	Cu	23–28	<4	<4
	Pb	2<	62–75	4–7
	Zn	1<	1–2	53–58
	Mixed	1.4	10–18	30–40
	ISP	–	1–6	45–50
Results obtainable from complex sulphide ores	Cu	17–25	3–7	3–11
	Pb	1–4	35–55	3–30
	Zn	0.5–3	1–5	45–55

Table C7. Sulfate processes for complex sulfide bulk concentrates

	Materials treated at present	*System*	*Main stages*	*Main products by-products rejects*	*Limitations/ problems*	*Level of development*
Sherritt Gordon[26]	Bulk concentrate 30% Zn 10% Pb 1% Cu Upgraded tailings 30% Zn 4% Pb 1% Cu From New Brunswick	Pressure leach sulphate/ electrowinning	H_2SO_4 pressure leach with return tankhouse electrolyte Iron removal Purification Chloride removal cadmium recovery Electrowinning	*Zinc* (Special High Grade) Cadmium Silver/lead sulphate Copper cake Pyrite Elemental sulphur Iron oxide	Fluoride—chloride removal; re-leaching for Pb/Ag. Recovery of Pb/Ag; residue low-grade	Continuous laboratory scale
New Brunswick Research and Productivity Council[28]	Bulk concentrate 30% Zn 1–5% Cu 5–15% Pb Upgraded tailings 20–30% Zn 1% Cu 2–4% Pb From New Brunswick	Roast leach sulphate/ electrowinning	Roasting Dilute H_2SO_4 leach with return tankhouse electrolyte; hot strong H_2SO_4 leach of residues recycled to reactor; solvent extraction/electrowinning of copper. Solution purification involves iron removal, cobalt cementation, cadmium recovery, electrowinning of zinc and brine leach of leach residue for recovery of Pb and Ag	*Zinc* (Special High Grade) Cadmium Cobalt Lead oxide/silver H_2SO_4 Iron oxide	Zinc production dependent on acid market. High capital cost	Continuous mini-planting
Técnicas Reunidas Comprex process/ Zincex process[27]	Bulk concentrate (from Spain) 28% Zn 3% Cu 6% Pb 400 g/t Ag Pyrite	Pressure leach solvent extraction—sulphate/ electrowinning	Leaching: 200°C, 7–14 kg/cm^2; O_2 pressure; 2 h; 10% solids All sulphides oxidized to sulphates, iron hydrolysed and precipitated as α-Fe_2O_3 Fe_2O_3 filtered/washed, contains Pb + Ag; copper recovery from leach by LIX SX/electrowinning. Two-stage solvent extraction for Zn First-stage Zn extracted with amberlite LA_2 to remove Zn as $ZnCl_4$; zinc stripped in water Second-stage Zn extraction to convert chloride to sulphate with D2EHPA Careful pH adjustment with ammonia or lime. Stripping carried out with spent electrolyte from zinc sulphate electrowinning	Zinc Copper Lead Silver Cadmium Fe_2O_3 Gypsum Waste water (Na^+ and Cl^-) Cl and Na	If copper content in conc. is high, Cu extraction circuit may be bottleneck in continuous circuit	Pilot plant: 1 t/h leach plant for total bulk concentrate Zincex process: commercial; 8000 and 11 000 t/year zinc

NOTE: This and the following table are from Barbery, Fletcher and Sirois, 1980.

Table C8. Chloride hydrometallurgical processes for bulk concentrates

Name	*Leaching*		*Metals separation stages*	*Metals production*	*Reagent regeneration*	*Effluent*	*Stage of development*	*Remarks*
	Reagents, g/l	*Conditions*						
CLEAR, U.S.A.[28]	Cu^{2+} 50–60 NaCl 200 Air/O_2 Cement or scrap Cu	Two stages (1) Reduction at 107°C (2) Oxidation with O_2 at 140°C	Details for Zn, Ag, etc., not disclosed	Direct electrowinning of Cu/Cl in diaphragm cell	Anodic oxidn. of Cu^+ to Cu^{2+}; air oxidation of Fe^{2+} to Fe^{3+}	Leach residue contains pyrite and S^0; cyclic process	Pilot plants operated: commercial plant, 100 t/d	Probably most advanced of chloride processes
Minemet, France[29]	Cu^{2+} 50 NaCl 200	Two stage countercurrent at boiling point	$PbCl_2$ cryst. Ag cement Fe goethite Zn, Cu } DEHPA SX	Cu and Zn electrowin from SO_4 soln	Air oxidation during Fe removal and SX	Leach residue contains pyrite + S^0; cyclic process	Litre-scale pilot plant	Pyrite not attacked. Air oxidation of Cu^+ to Cu^{2+} during SX controls pH
Zincex, Spain[30]	Cl + SO_4 40	Leaching of calcine from salt roasting of pyrite cinders	Zn SX in two stages (1) R NH_2 Cl water strip (2) DEHPA, NH_3 or CaO for pH adjust. Cu SX with LIX?	Zn electrowin. from SO_4 soln; probably Cu also	None—maybe some Cl recovery	Leach residue and SX; raffinate for disposal	Commercial plant in operation for Zn	Ag not leached so separate circuit necessary
Elkem, Norway[31]	Fe^{3+} 150 Cu^{2+} 20 NaCl up to 150	Two-stage countercurrent at boiling point	$PbCl_2$ cryst. Zn I EX or SX	Direct electro win. Cu and Zn from chloride soln	Fe^{2+} oxidation by air and Cl_2 from Zn electrolysis	Leach residue; cyclic process	Small pilot plant for Cu, and perhaps Zn	Ag recovered in Cu cathode
Dextec, Australia[32]	NaCl 200 Air electricity 1 kwh/kg Cu HCl	Anodic dissolution in presence of air at pH 2–2.5	None disclosed	Electrolytic deposition of Cu during leaching Anode and cathode of graphite	None. Could be recovery of Cl	Leach residue and probably bleed for impurity control	300-l pilot-scale cells operated. Diaphragm of polypropylene filter cloth	So far only application to copper disclosed. Could have more general application
Cymet, Cyprus Mines, U.S.A.[33]	Cu^{2+}, Fe^{3+}, NaCl } Close to saturation	Two-stage at boiling point; atmospheric pressure	Cool to cryst. Cu Cl	H_2 reduction of CuCl in fluid bed	Oxidation of Fe^{2+} with O_2 after first-stage leach	Cyclic process; leach residue and Fe ppt	200 lb/h pilot plant	Original process involved anodic dissolution
Haver and Wong[34] U.S.B.M.[35]	Fe^{3+} 212 HCl 20–100 $\frac{FeCl_3}{CuFeS_2} = 2.7$	At boiling point fine grind and gentle stirring	Cu—cement with sponge Fe $FeCl_2.4H_2O$. Cryst. S^0 and Ag recovered leach residue with perchlorethylene and cyanide	Cement Cu	$FeCl_2$ hydrolysis. HCl electrolysis and Cl_2 oxidation of Fe^{2+}	Cyclic process; residue discarded	Lab. scale development	One of first attempts at chloride process development. High energy requirement. Cu can be recovered by electrowinning
U.S.B.M.[36]	Cl_2 O_2	Glass-lined autoclave at 100°C and 50 lb/in²	SX for Cu $ZnCl_2$ cryst. $PbCl_2$ cryst. Ag cement with Pb	Zn and Pb by molten salt electrolysis. Cu could be by electrowinning. Cement Ag	Cl_2 regenerated from molten salt electrolysis	Leach residue contains S^0	Lab. pilot plant	Evaporation of Zn Cl_2 soln necessary. Molten salt electrolysis is new development. Low energy requirement

(From Cases, 1980)

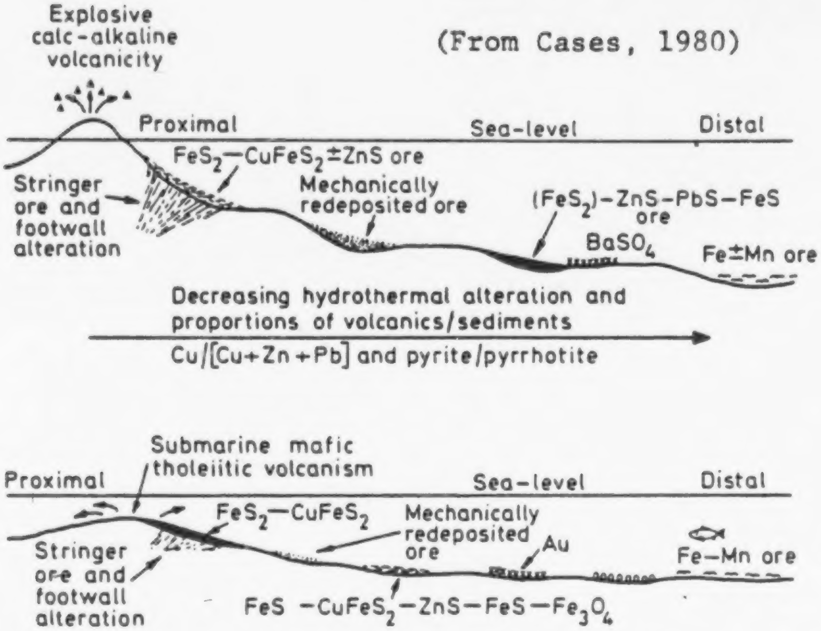

Explosive calc-alkaline volcanicity
Proximal
Sea-level
Distal
FeS$_2$—CuFeS$_2$±ZnS ore
Stringer ore and footwall alteration
Mechanically redeposited ore
(FeS$_2$)-ZnS-PbS-FeS ore
BaSO$_4$
Fe±Mn ore
Decreasing hydrothermal alteration and proportions of volcanics/sediments
Cu/[Cu+Zn+Pb] and pyrite/pyrrhotite
Submarine mafic tholeiitic volcanism
Proximal
Sea-level
Distal
FeS$_2$—CuFeS$_2$
Mechanically redeposited ore
Au
Stringer ore and footwall alteration
Fe—Mn ore
FeS —CuFeS$_2$—ZnS—FeS—Fe$_3$O$_4$

Fig. B1. Genesis of complex sulphide ores. After Plimer[7]

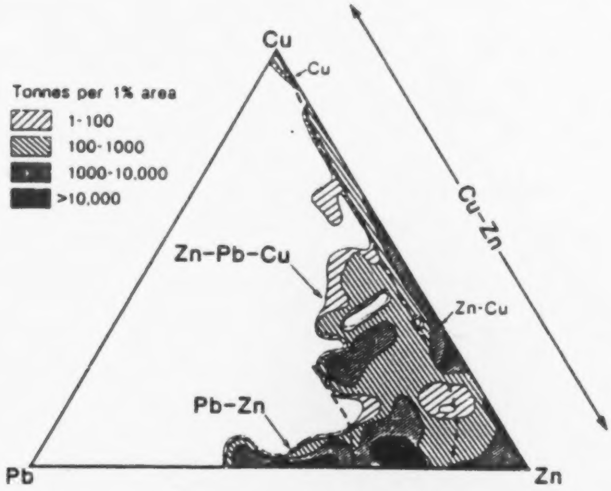
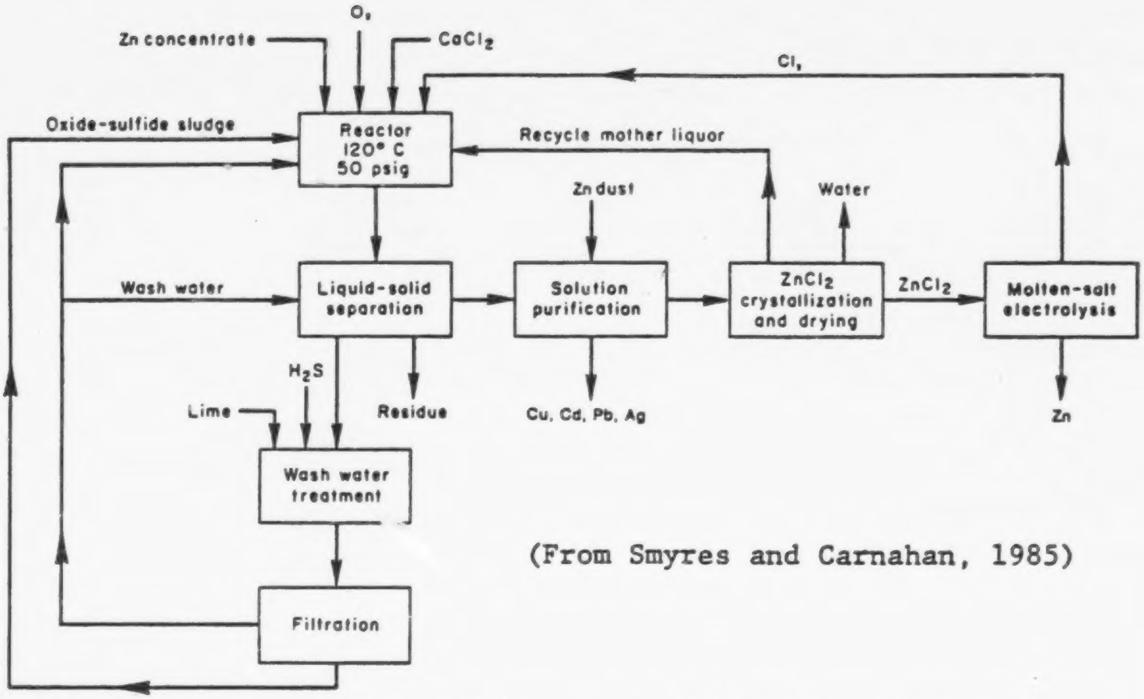


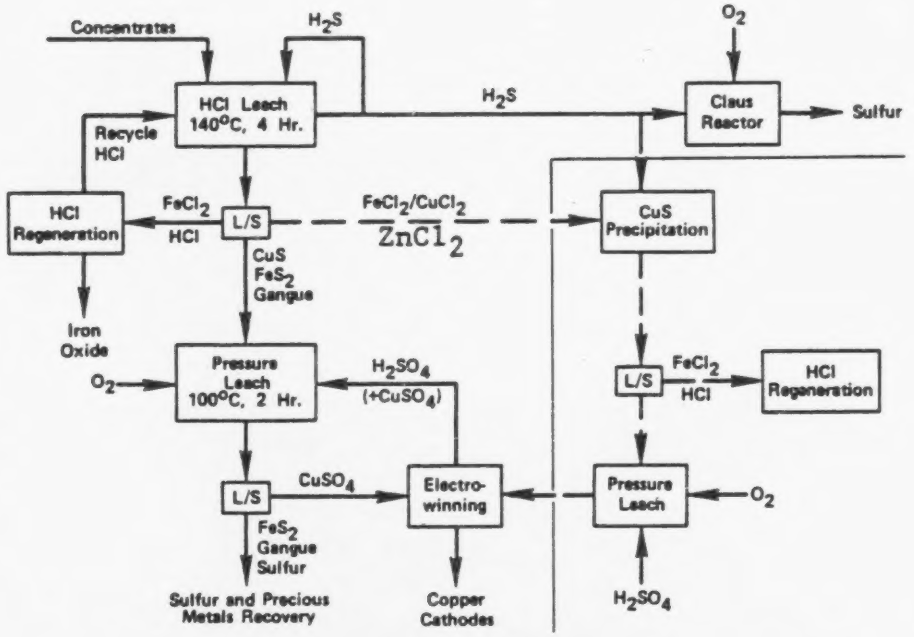
Fig. B2. Metric tons of contained Cu, Pb, and Zn in massive sulfide deposits from the Canadian Shield, Scandinavian Caledonides, Green Tuff belt, Japan; Bathurst, New Brunswick; Iberian pyrite belt; Fennoscandian Shield districts; and all of the principal sediment-hosted deposits (Large, 1980) contoured to illustrate the three principal compositional groups. The volcanic-associated deposits are entirely in the Cu-Zn and Zn-Pb-Cu groups; the Pb-Zn group is largely in "sediment-hosted" deposits.

(From Franklin, Lydon, and Sangster, 1981)


Zn concentrate
O_2
$CaCl_2$
Cl_2
Oxide-sulfide sludge
Reactor 120° C 50 psig
Recycle mother liquor
Zn dust
Water
Wash water
Liquid-solid separation
Solution purification
$ZnCl_2$ crystallization and drying
$ZnCl_2$
Molten-salt electrolysis
H_2S
Lime
Residue
Cu, Cd, Pb, Ag
Zn
Wash water treatment
Filtration

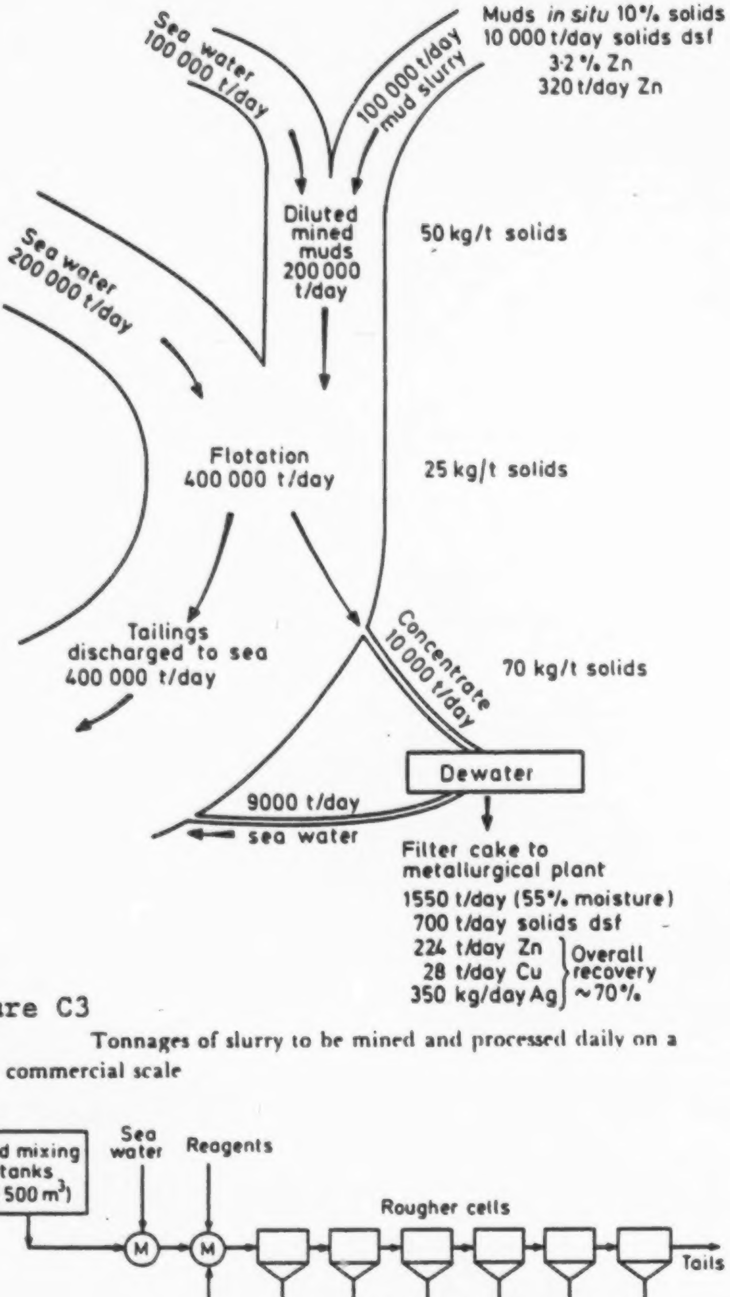
(From Smyres and Carnahan, 1985)

Figure C1. Conceptual flowsheet for chlorine-oxygen leaching of low-grade zinc concentrate.


Concentrates
H_2S
HCl Leach 140°C, 4 Hr.
H_2S
O_2
Claus Reactor
Sulfur
Recycle HCl
HCl Regeneration
$FeCl_2$
HCl
L/S
$FeCl_2/CuCl_2$
$ZnCl_2$
CuS Precipitation
CuS
FeS_2
Gangue
Iron Oxide
O_2
Pressure Leach 100°C, 2 Hr.
H_2SO_4
$(+CuSO_4)$
L/S
$FeCl_2$
HCl
HCl Regeneration
L/S
$CuSO_4$
Electrowinning
Pressure Leach
O_2
FeS_2
Gangue
Sulfur
Sulfur and Precious Metals Recovery
Copper Cathodes
H_2SO_4

(From Harvey and Dudas, 1977)

Figure C2. Simplified flow diagram for a hydrometallurgical process based on nonoxidizing HCl pressure leaching of copper concentrates.


Sea water 100 000 t/day
100 000 t/day mud slurry
Muds *in situ* 10% solids
10 000 t/day solids dsf
3·2 % Zn
320 t/day Zn
Diluted mined muds 200 000 t/day
50 kg/t solids
Sea water 200 000 t/day
Flotation 400 000 t/day
25 kg/t solids
Tailings discharged to sea 400 000 t/day
Concentrate 10 000 t/day
70 kg/t solids
Dewater
9000 t/day
sea water
Filter cake to metallurgical plant
1550 t/day (55% moisture)
700 t/day solids dsf
224 t/day Zn
28 t/day Cu
350 kg/day Ag
Overall recovery ~70%

"dsf" = dry, salt-free

(Figures from Fletcher and Mustafa, 1980)

Figure C3

Tonnages of slurry to be mined and processed daily on a commercial scale

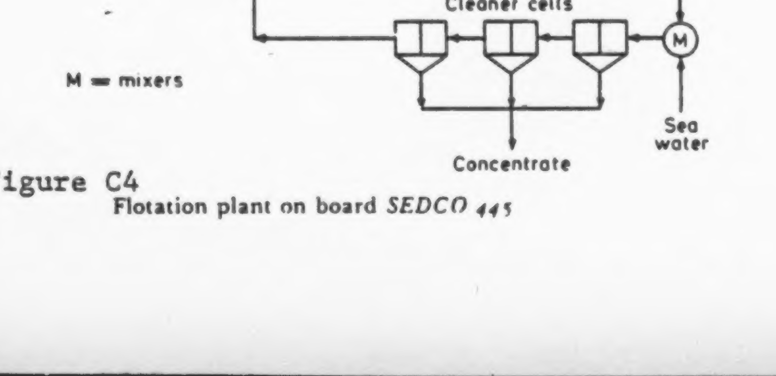
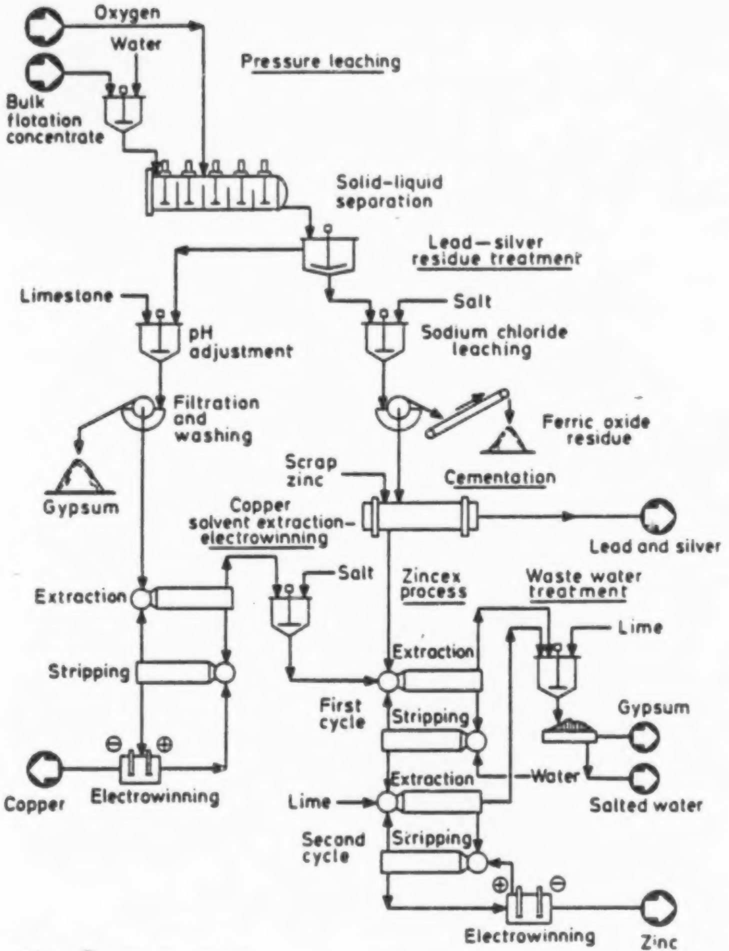

Mud mixing tanks (500 m³)
Sea water
Reagents
Rougher cells
M
M
Tails
Cleaner cells
M
M = mixers
Sea water
Concentrate

Figure C4

Flotation plant on board *SEDCO 445*


Oxygen
Water
Pressure leaching
Bulk flotation concentrate
Solid-liquid separation
Lead—silver residue treatment
Limestone
Salt
pH adjustment
Sodium chloride leaching
Filtration and washing
Ferric oxide residue
Gypsum
Scrap zinc
Cementation
Copper solvent extraction–electrowinning
Lead and silver
Salt
Zincex process
Waste water treatment
Extraction
Lime
Stripping
Extraction
First cycle
Stripping
Gypsum
⊖
⊕
Extraction
Water
Copper
Electrowinning
Lime
Salted water
Second cycle
Stripping
⊕
⊖
Electrowinning
Zinc

(From Nogueira et al., 1980)

Figure C5. Comprex process

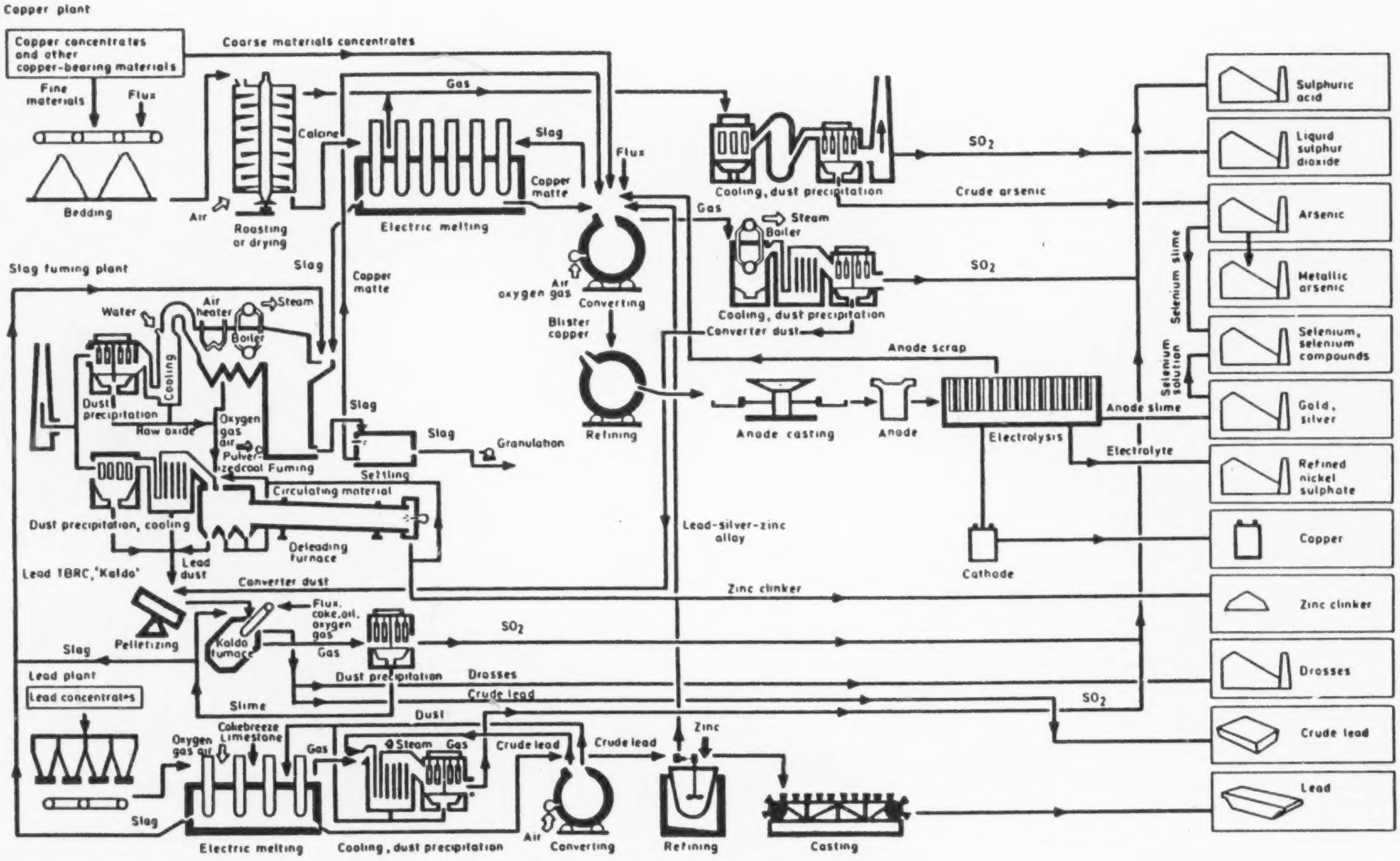

Copper plant
Copper concentrates and other copper-bearing materials
Coarse materials concentrates
Fine materials
Flux
Bedding
Air
Roasting or drying
Calcine
Gas
Slag
Flux
Copper matte
Electric melting
Air oxygen gas
Converting
Blister copper
Refining
Cooling, dust precipitation
SO2
Crude arsenic
Gas
Steam
Boiler
Cooling, dust precipitation
Converter dust
Anode scrap
Anode casting
Anode
Electrolysis
Anode slime
Electrolyte
Cathode
Selenium slime
Selenium solution
Sulphuric acid
Liquid sulphur dioxide
Arsenic
Metallic arsenic
Selenium, selenium compounds
Gold, silver
Refined nickel sulphate
Copper
Zinc clinker
Drosses
Crude lead
Lead
Slag fuming plant
Slag
Copper matte
Water
Air heater
Steam
Boiler
Cooling
Dust precipitation
Raw oxide
Oxygen gas air
Pulverizedcoal
Fuming
Slag
Settling
Slag
Granulation
Circulating material
Dust precipitation, cooling
Deleading furnace
Lead dust
Lead-silver-zinc alloy
Lead TBRC, 'Kaldo'
Converter dust
Zinc clinker
Pelletizing
Flux, coke, oil, oxygen gas
Kaldo furnace
SO2
Gas
Slag
Dust precipitation
Drosses
Crude lead
Slime
SO2
Lead plant
Lead concentrates
Dust
Oxygen gas air
Cokebreeze Limestone
Gas
Steam
Gas
Crude lead
Crude lead
Zinc
Slag
Electric melting
Cooling, dust precipitation
Air
Converting
Refining
Casting

Fig. C6. Boliden's Ronnskar works: flow chart from Barbery, Fletcher and Sirois, 1980.

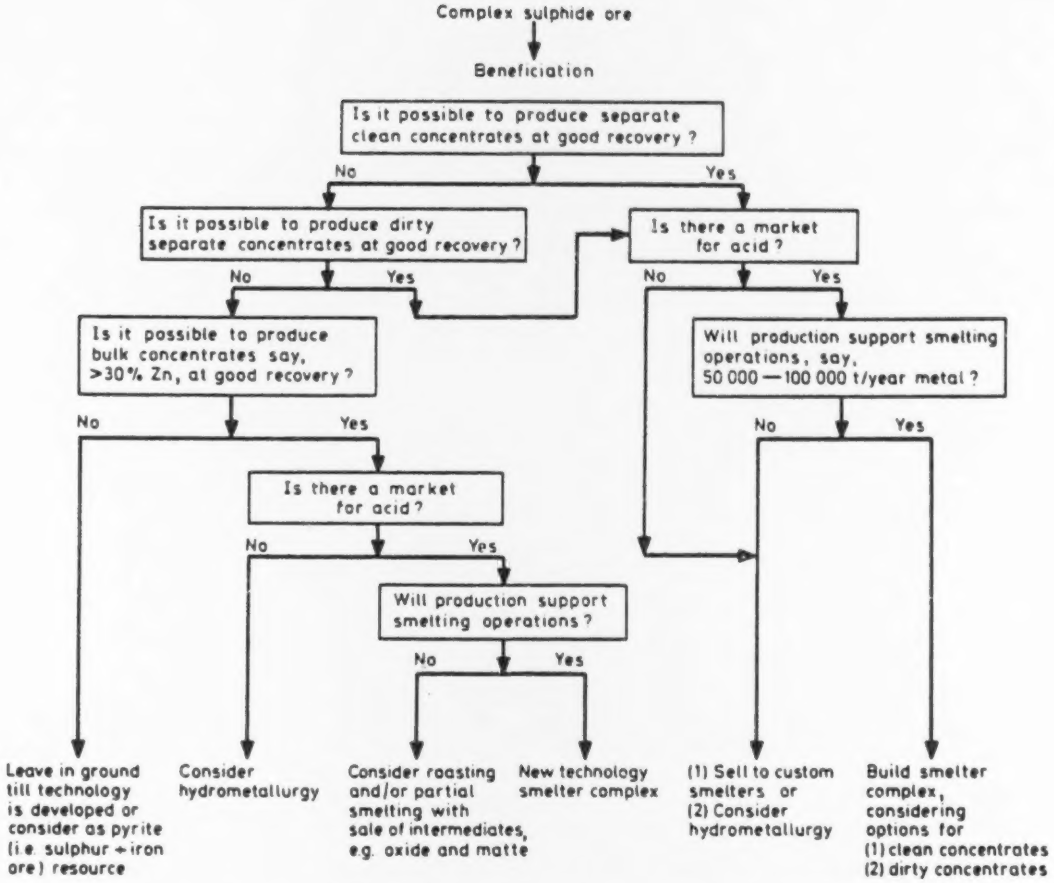

Complex sulphide ore
Beneficiation
Is it possible to produce separate clean concentrates at good recovery ?
No
Yes
Is it possible to produce dirty separate concentrates at good recovery ?
Is there a market for acid ?
No
Yes
No
Yes
Is it possible to produce bulk concentrates say, >30% Zn, at good recovery ?
Will production support smelting operations, say, 50 000 — 100 000 t/year metal ?
No
Yes
No
Yes
Is there a market for acid ?
No
Yes
Will production support smelting operations ?
No
Yes
Leave in ground till technology is developed or consider as pyrite (i.e. sulphur + iron ore) resource
Consider hydrometallurgy
Consider roasting and/or partial smelting with sale of intermediates, e.g. oxide and matte
New technology smelter complex
(1) Sell to custom smelters or (2) Consider hydrometallurgy
Build smelter complex, considering options for (1) clean concentrates (2) dirty concentrates

Fig. C7. Decision tree for processing routes for complex sulphides (From Barbery et al., 1980)

III. Oxide Nodules and Crusts

A. Broad Intercomparisons

Without intent to trivialize the vast amount of information bearing on the distinctions between ferromanganese oxide nodules and crusts, let it be said that they appear to be at least first cousins. Thus, it is appropriate to compare average properties and tendencies. For example, the often pointed-out tendency for high grade nodules of the Pacific to occur on abyssal plains, in contrast to major findings of high grade crusts on the slopes of seamounts and the like, does not preclude occurrence in other subsea settings. A reported finding of cobalt-rich (to 0.6% Co) nodules at 5000-6000 m water depth in the western Pacific (information attributed to D.S. Cronan, 17th UMI, Biloxi, 1986) further illustrates the overlap of resource characteristics.

In normal metal markets, high grade nodules would be considered a potential nickel ore, with major by-product copper and cobalt; the so-called cobalt-rich crusts, based on information to date, would be considered a potential cobalt ore with major by-product nickel and minor by-product copper. In essence, the relative importance of nickel and cobalt is reversed in the classes of nodules and crusts that are of greatest current interest for eventual commercial mining.

Cobalt's historically higher value compared with nickel and perceived greater vulnerability to supply disruption and price run-up are key factors in a shift of interest in some quarters from nodules to crusts. Another key factor is the location of potentially large resources of cobalt-rich crusts within exclusive economic zones in the Pacific Ocean. Whether relative ease of mining is also favorable for crusts remains to be demonstrated.

Figure 10 is a representation of the overall chemical constitution of cobalt-rich crusts, that was constructed from published and inferred average compositions as of mid-1983. Several features are worthy of special note:

- the prominent presence of contained water: 40% or more by weight or about 2/3rds of the volume of superficially (drained) dry crust
- the corresponding presence of sea salts, giving rise to a significant chloride and alkali metal content that must be reckoned with in mineral processing and extractive metallurgical (MP&EM) treatments
- two percent or so of value-metal oxides, contained mainly in the iron and manganese oxide phases and not amenable to concentration by physical beneficiation techniques .

The above features and the underlying extreme fineness of the grain and pore structure are qualitatively the same as for the nodules. However, by virtue of their gross physical condition, as discrete bodies resting on unconsolidated sediments, proper sampling and analysis of nodules gives direct indication of prospective ore composition. Gangue constituents are contained mainly within the nodules. This is not generally true of crusts which, to varying degrees, adhere to and are intergrown with substrate whose composition and consistency, moreover, are variable. One is, as a consequence, apt to be misled in basing mining, processing and profitability considerations on analyses of crust with no adhering or occluded substrate material.

Piled up on land, both nodules and crusts would be considered attractive ores. At present, perceived advantages of crusts--in particular, higher average cobalt content, shallower depth and EEZ location--are offset by the lack of efficient and selective mining techniques. This may well be a more urgent concern than devising suitable processing technology. If it proves not feasible to mine crusts without also carrying along more than a reasonably small percentage of substrate (some substrate materials are more deleterious than others, in the MP&EM sequence), an inexpensive separation scheme will be required, at minimum.

B. Major Processing and Marketing Issues

The important issues in processing and marketing have to do with realizable ore values and with the existence and condition of receiver industries and supporting infrastructure. In the former category, attention is called to the very large amplitude of swings in the free-market price of cobalt; the possible long-term overcapacity (market economies) in primary nickel, as affected by the strong sensitivity of nickel production costs from laterites to fuel prices; and the question of production of manganese.

Interesting platinum occurrences (to about 1 ppm) have been reported for cobalt-rich crusts. Whether platinum content will prove to be economically significant will depend upon a number of factors, including its mineralogy and distribution among the major mineral phases of the crusts. Recoverability of the platinum, as well of other potentially valuable by-products, will also be affected by the choice of metallurgical process.

Major iron oxide content of nodules and crusts has generally not been considered as a potential mineral resource, except for iron which would accompany manganese in the production of ferromanganese. On the other hand, success has been reported for the production of steel from iron oxide concentrate that was derived from Nicaro, Cuba nickel plant tailings (BuMines, 1984). The Nicaro process entails reduction roasting and ammoniacal leaching of lateritic nickel (oxide) ore of the high-iron variety. There may, then, be specific circumstances in the exploitation of ferromanganese oxide nodules or crusts whereby it would make sense to produce at least a portion of the iron separately from the manganese.

The salient features of the supply and price of primary cobalt, from the U.S. standpoint, are its unfavorable geographical distribution and projected long-term overcapacity. Essentially, the African nation of Zaire has about half of the market-economy cobalt capacity, and Zaire and its neighbor, Zambia, together account for about two thirds. Additional amounts of cobalt are produced in Zimbabwe, Botswana and South Africa, making for a concentration of production capacity in southern Africa, as detailed by Hale (1983).

Market-economy country reserves and production capacity of manganese (and of chromium, but not of nickel) are similarly concentrated in southern Africa, where production costs for manganese and cobalt (and chromium) have generally been lower than for other producers. Barring major new finds elsewhere, the distribution is likely to become more skewed in the long term, as high-grade reserves in other parts of the world become depleted. Accordingly, for a new supply of cobalt, nickel and manganese to compete in world markets, total costs (mining, processing and transportation), must be competitive with those of the current low-cost major producing areas. In this connection, it is worth noting that substantially all primary cobalt production is as a by-product of copper and nickel mining, with mining costs assignable to the major metal and with the option available, within limits, to defer cobalt production pending more favorable prices.

Having in mind these nickel and cobalt market considerations, a number of evaluations of economic potential of manganese nodules and, more recently, crusts have included the provision that revenues from the sale of manganese would be required in order to yield a positive cash flow. The demand for manganese metal being relatively limited, the thinking has focused on production of ferromanganese. The appropriate waste or sidestream from the processing of crusts or nodules ought, then, to be competitive with imported, high grade manganese ore as a source for ferromanganese manufacture. However, it is not assured that the incremental revenue from sales of ferromanganese from nodules or crusts would exceed the incremental cost of production (Harvey and Ammann, 1985).

A leach residue from a hydrometallurgical process designed for selective nickel and cobalt recovery would typically be of low grade as a manganese ore, in terms of manganese content, manganese-to-iron ratio (low), and content of deleterious impurities (high). The same observations apply to a manganese slag from a smelting process where, in addition, the supposed energy-advantage of starting with manganese-iron oxide in the molten state would probably be canceled out by the added difficulty of providing reaction interface between molten slag and low-density, solid carbonaceous reductant.

A more appropriate business strategy with respect to manganese from nodules or crusts would be to adjust that fraction of the manganese content which is recovered, to the market for higher value manganese products. Bulk products of relatively high price per unit of contained manganese include manganese chemicals (sulfate and others), electrolytic manganese dioxide and electrolytic manganese metal. For a hydrometallurgical process, this strategy implies adjusting process conditions so as to result in the desired net solubilization of the manganese in the nodules or crusts. Some possible routes to higher value products are illustrated in Figure 11 for a sulfuric acid pressure leach treatment.

A still broader issue than manganese production strategy concerns where to process and whether to refine the major values from nodules and crusts. A large metallurgical complex for the purpose would require a deep-water port and facilities, a large amount of reasonably level land and inexpensive electrical power (or fuel), process water and raw materials, as well as supporting infrastructure. U.S. sites fully meeting these requirements, within a relatively short transportation distance from promising nodule and crust mining areas in the Pacific, are clearly very few. In addition,one is obliged to consider a greenfield plant for which capital recovery, even for a location on the continental U.S., would pose an unacceptable economic burden in normal metal markets.

The alternative business strategy has been suggested of doing the primary processing in an island location such as Hawaii and shipping the process intermediates for refining elsewhere (Figure 12). The U.S. no longer has major nickel and cobalt refining capacity, nor is its residual ferromanganese capacity situated where processing of slag from Pacific Ocean nodules and crusts would make sense. Japan currently appears to be the most advantageously poised, in terms of location and also of experience and existing capacity for processing and refining imported mineral raw materials. These advantages have been somewhat dulled by the considerable appreciation of the yen, and it is conceivable that custom cobalt and nickel refining could eventually be more cheaply carried out in some other Pacific Rim nation.

In considering the possibility of having process intermediates refined elsewhere, by other than the primary processor, it needs to be recognized that the value received for the nickel and cobalt might be only 60% or so of their value as commercial product forms. If announced plans to mine and process nickel laterite deposits of SW Oregon and NW California are followed through, there would be the possibility of converting the metallurgical plant to crusts or nodules. The timing of the possible changeover would be compatible with the limited reserves of nickel laterites, on the one hand, and the decade or more needed to bring an ocean hard minerals mining industry into being, on the other.

C. Recent Process-Related Studies

The scale-back of industrial activities in manganese nodule processing R&D, beginning about a decade ago, has left a number of important issues unresolved. Continued uncertainty over economic recoverability of manganese, as discussed above, is one example. Few of the various metallurgical sequences proposed for nodules were carried out on a pilot scale in a continuous manner from ore to commercial product forms. At least partly out of reluctance to engage in activities perceived as more appropriately the concern of the private sector, interim process studies with support from the federal government have tended to be rearrangements or elaborations of information and inferences previously arrived at. Meanwhile, industry-supported process development has been meager. Development of prototype mining systems for nodules appears to have fared no better.

The following account is mainly confined to highlights of recent laboratory studies of MP&EM of subsea polymetallic oxide nodules and crusts. The intent is to illustrate the scope of current activities in this area, rather than to provide a comprehensive survey.

At the November 20 OTA workshop, J.R. Hein summarized resource information obtained in the ongoing USGS program on ferromanganese oxide crusts. Average compositions by weight for crusts from <2500-meters water depth in the central Pacific, U.S. EEZ include 0.78% Co, 0.44% Ni, 0.08% Cu, 22% Mn, and Fe/Mn = 0.71. Cobalt and nickel content are reported to correlate with the manganese (predominantly δ-MnO_2) phase and copper with the iron ($FeOOH \cdot xH_2O$) phase, the former increasing in prominence with decreasing water depth. This updated information bears out the characterization of the crusts as a potential cobalt ore and also the contention that, with an average total nonferrous value metal content of only 1.3%, major dilution by substrate in the mining operation would result in less than marginal ore. See Figure 13 for value metal contents of world nickel laterite deposits and note that average % Ni+Co for crusts is comparable to that for the SE Oregon nickel mine (Riddle) and the NW California nickel prospect (Gasquet Mountain).

D.G. Foot and co-workers (1986) at the U.S. Bureau of Mines Salt Lake City Research Center have been working on the separation of co-mined substrate by flotation. Column flotation, illustrated in Figure 14, has been shown to be effective and superior to conventional flotation for the indicated purpose. Whether column flotation would operate with adequate efficiency on board ship, as suggested by Foot et al., would need to be demonstrated. Industrial columns range up to 13 m in height and 0.3 to 1.8 m in cross section (Dobby and Finch, 1986). Access to a ship's motion simulator, such as that employed at the Warren Spring Laboratory (U.K.) to evaluate floatability of Red Sea muds, would be useful at this stage.

The Bureau of Mines flotation test set-up features external bubble generation, with injection of a pressurized air/water mixture into the column. It is interesting that, in achieving about 90% recovery of the value metals in a flotation concentrate, copper as well as nickel and cobalt followed the manganese. Of probably greater significance in terms of overall process economics, since copper is a minor value in the crusts, is the observed higher reagent consumption when sea water was used in place of fresh water in column flotation (Foot, November 20 OTA workshop). The reported results on mineral processing are encouraging. Development of the ability to build a high degree of selectivity into the mining system would be still more advantageous, however.

Preliminary work on hydrometallurgical processing of crust samples and flotation concentrates has also been carried out at the Salt Lake City Research Center. Two approaches were found to result in high extractions of cobalt and nickel during the leach step. The first of these, leaching with aqueous SO_2 under ambient conditions, is known but is not widely favored for nodules because of high solubilization of the manganese and iron. The test samples of crust, analogous in composition to nodules, reacted in the expected way. The second approach has not been commonly mentioned for nodules. It involves reduction roasting, followed by oxygen pressure leaching in dilute H_2SO_4, and gave selectively high rejection of the iron. It is not apparent what advantages this second approach has over reduction roast / ammoniacal leach on the one hand and sulfuric acid (concentrated H_2SO_4) pressure leach on the other. That information will, no doubt, be revealed in follow-up studies of metals separation and recovery from the leach solutions.

Parallel research on the processing of cobalt-rich crusts from the Hawaiian EEZ has recently been completed at the Bureau's Avondale Research Center (Haynes, November 20 OTA workshop). The study included chemical and mineralogical characterization of crust samples and characterization of tailings produced in the laboratory by three different metallurgical treatments. As in the prior comprehensive experimental study of nodules processing (five different metallurgical routes), the process wastes were found to be nonhazardous as defined by RCRA-specified test procedures.

The preliminary metallurgical studies on cobalt-rich crusts at Avondale included "Cuprion" (cuprous-ion mediated, aqueous CO reduction) ammoniacal leaching, sulfuric acid pressure leaching, and reduction smelting (Haynes et al., 1987). The two leach processes selected have the advantage of not requiring a drying step. Consistent with the results of various studies on nodules and with the expectations expressed by Harvey and Ammann (1985), percent extraction of cobalt, nickel and copper and rejection (i.e., nonleaching) of manganese and iron were high in sulfuric acid pressure leaching. By contrast, ammoniacal leaching resulted in only fair extraction of nickel and fair rejection of manganese and iron and, as expected, poor (about 50%) extraction of cobalt.

The laboratory evaluation of matte smelting was apparently carried only through the initial reduction-smelting step. The resulting high (≥96%) recoveries of cobalt, nickel and copper in the high-iron ferroally button and the high (ca. 99%) rejection of the manganese into the silicate slag reflect good technique but have limited process significance. The subsequent steps in matte smelting, namely, oxidizing and slagging-off of the manganese and the bulk of the iron of the molten ferroalloy and sulfiding the metal phase to matte, afford opportunities for significant losses of nonferrous metal values.

Loss mechanisms in matte smelting include entrainment in the slag, solubility of sulfide (e.g., Cu_2S) in the slag, and reoxidation (especially, Co to CoO) and reporting of the oxide to the slag. The question of potentially low recovery of the major metal should be of broad concern, i.e., not only to those having interest in metallurgical details. Low yield in processing could have a more adverse effect on economic feasibility than dilution of the ore grade by substrate. Information bearing on unresolved parts of this question is suggested by surveys and test results on nickel laterite processing.

Expected recoveries by three metallurgical processes are given (next page) for the Buhinda, Burundi nickel deposit which grades, at 0.8% Ni cutoff, 1.62% Ni, 0.12% Co and 0.31% Cu (Figure 13). According to information from the Burundi Ministry of Public Works, Energy and Mines (1986), nickel, cobalt and copper recoveries in the crude matte from the furnace are all high (about 95% each). In the matte conversion step, however, a loss of more than half of the contained cobalt may be experienced owing to the oxidizing atmosphere (ease of oxidation of the metals decreases in the order Fe, Co, Ni, Cu).

Sulfuric acid pressure leaching appears to offer the highest yields of cobalt and nickel, of the processes that have been worked out and published in reasonable detail. Pilot plant development programs by AMAX-COFREMMI on a variety of laterite ores from around the world demonstrated routine recoveries in excess of 90%, with low energy consumption and nearly stoichiometric (i.e., small excess) acid consumption (Wicker and Jha, 1986). For adaptation of the process to cobalt-rich crusts, M.C. Jha (personal communication, 1986) of AMAX recommends a higher operating temperature of 270 °C and staged acid addition as well as use of compartmented horizontal autoclaves. On the last point, Sarkar (1985) concluded that a series of vertical autoclaves is more economic on account of its higher availability. Supplemental AMAX recommendations of staged flashing (steam pressure release) and energy recovery were comprehended in the Harvey and Ammann study (1985), in which the leach temperature was taken to be 255-260 °C.

Among the entirely different approaches to processing that have been proposed recently for deep-sea ferromanganese oxides, three will be mentioned in closing, without elaboration. One approach, from Argonne National Laboratory, involves extraction and selective volatilization by a molten chloride salt (NaCl, KCl, $MgCl_2$ eutectic mixture at 450 °C) and electrodeposition of cobalt and manganese metals from the resulting molten salt solution. A second approach (CSA, Inc.), applicable more broadly to low-grade manganese ores, involves reductive dissolution in concentrated formic acid solution, metals separation by counter-current ion exchange, and recovery by electrowinning from chloride solution with continuous acid removal. A third approach (Cambridge University) entails hydrochloric acid-mediated reduction of the MnO_2 matrix by powdered coal.

Manipulation of the well known reaction

$$MnO_2(s) + 2HCl(aq) = MnCl_2(aq) + Cl_2$$

is a feature of formic acid and powdered coal reductive dissolution processes as well as of the earlier hydrochloric acid dissolution process of Metallurgie Hoboken Overpelt. MHO may have extended their earlier work and, in general, HCl hydrometallurgy continues to be a candidate for processing nodules and crusts.

Much remains to be done in the important area of process definition, and it is to be hoped that recent work will continue and be extended.

Table (p. 10) Expected metal recoveries in the processing of Burindi nickel laterites (E&MJ, July 1986, p.9)

	Ni	Cu	Co
Reduction Roast/Ammoniacal Leach	73-78	70-75	38-50
Sulfuric Acid Pressure Leach	90	90	90
Matte Smelting	93-96	94-97	30-40

D. Figures, Section III

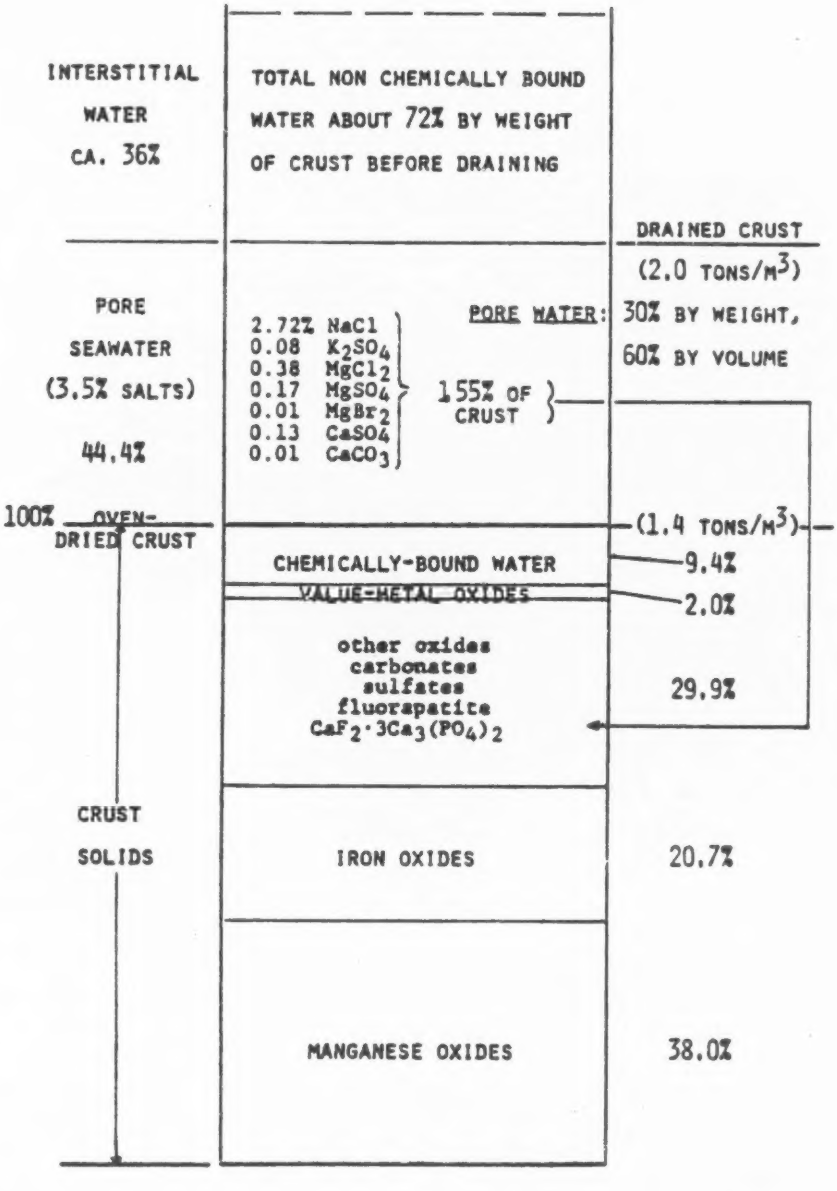

INTERSTITIAL WATER CA. 36%
TOTAL NON CHEMICALLY BOUND WATER ABOUT 72% BY WEIGHT OF CRUST BEFORE DRAINING
DRAINED CRUST
(2.0 TONS/M^3)
PORE SEAWATER (3.5% SALTS) 44.4%
2.72% NaCl
0.08 K_2SO_4
0.38 $MgCl_2$
0.17 $MgSO_4$
0.01 $MgBr_2$
0.13 $CaSO_4$
0.01 $CaCO_3$
155% OF CRUST
PORE WATER: 30% BY WEIGHT, 60% BY VOLUME
100% OVEN-DRIED CRUST
(1.4 TONS/M^3)
CHEMICALLY-BOUND WATER
9.4%
VALUE-METAL OXIDES
2.0%
other oxides carbonates sulfates fluorapatite $CaF_2 \cdot 3Ca_3(PO_4)_2$
29.9%
CRUST SOLIDS
IRON OXIDES
20.7%
MANGANESE OXIDES
38.0%

Figure 10. Gross, metallurgically-significant features of the assumed crust composition.

(From Harvey and Ammann, 1985)

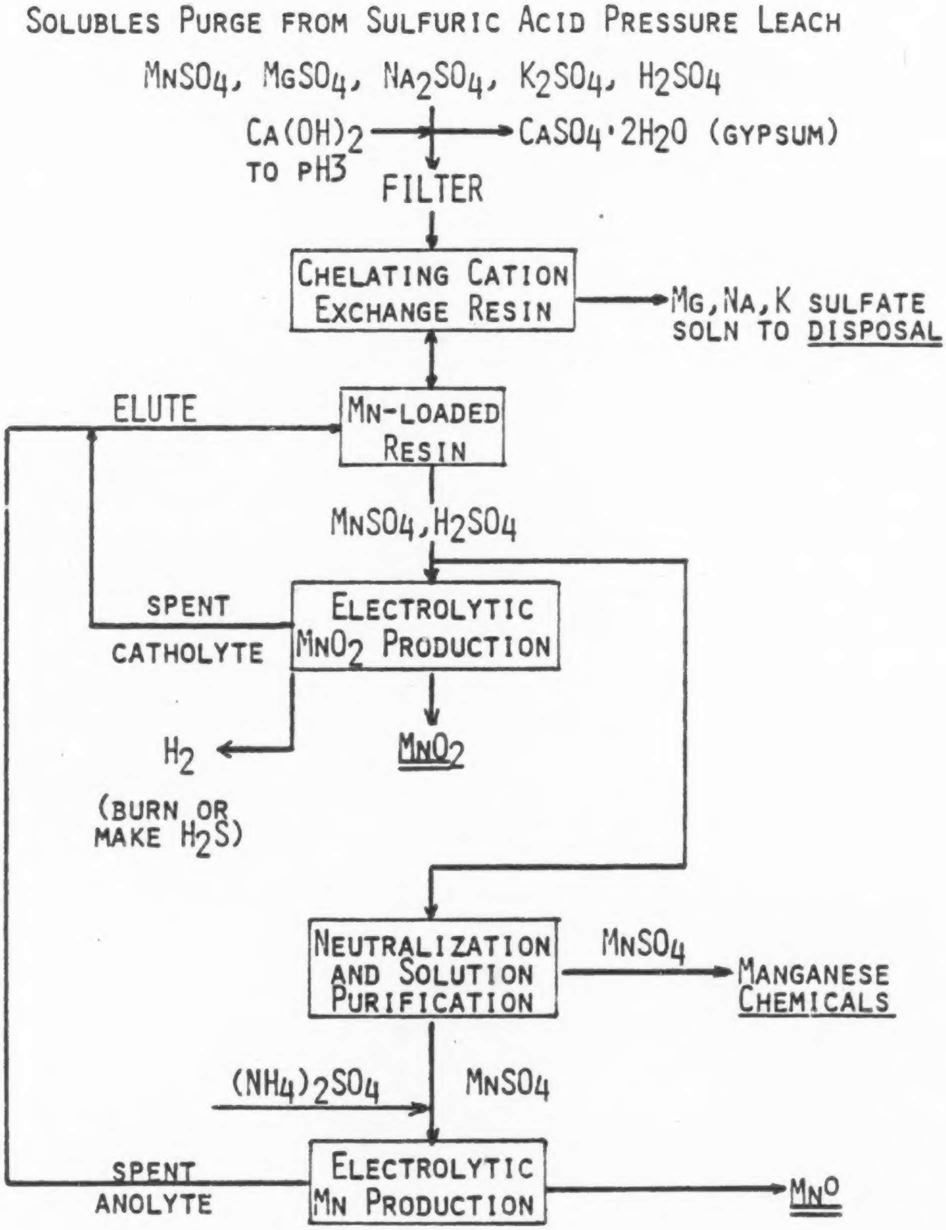

SOLUBLES PURGE FROM SULFURIC ACID PRESSURE LEACH
MnSO4, MgSO4, Na2SO4, K2SO4, H2SO4
Ca(OH)2 TO pH3
CaSO4·2H2O (GYPSUM)
FILTER
CHELATING CATION EXCHANGE RESIN
Mg, Na, K SULFATE SOLN TO DISPOSAL
ELUTE
Mn-LOADED RESIN
MnSO4, H2SO4
SPENT CATHOLYTE
ELECTROLYTIC MnO2 PRODUCTION
H2
(BURN OR MAKE H2S)
MnO2
NEUTRALIZATION AND SOLUTION PURIFICATION
MnSO4
MANGANESE CHEMICALS
(NH4)2SO4
MnSO4
SPENT ANOLYTE
ELECTROLYTIC Mn PRODUCTION
Mn°

Figure 11. Manganese production options for hydrometallurgical route.

(From Harvey and Ammann, 1985)

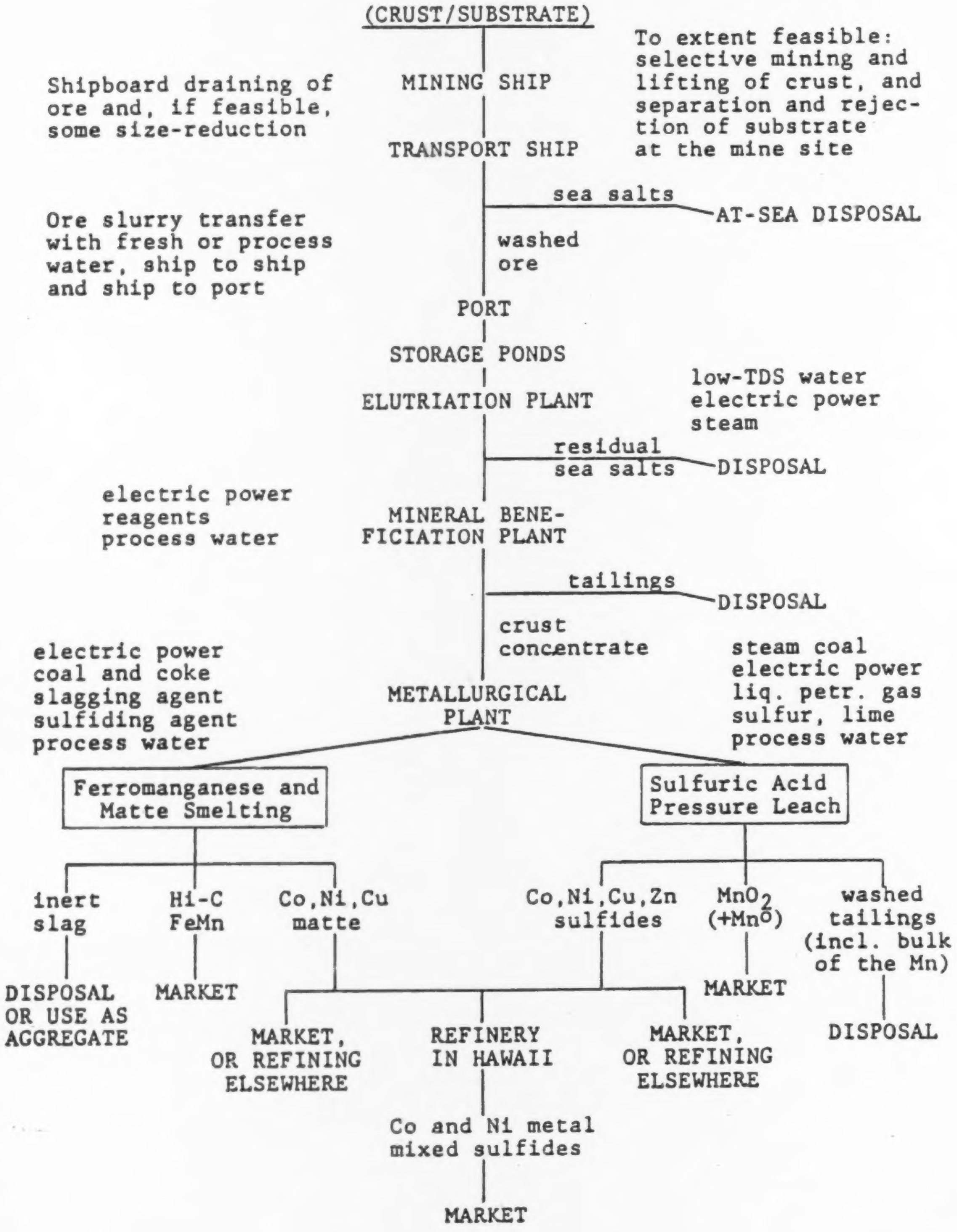

(CRUST/SUBSTRATE)
To extent feasible: selective mining and lifting of crust, and separation and rejection of substrate at the mine site
Shipboard draining of ore and, if feasible, some size-reduction
MINING SHIP
TRANSPORT SHIP
sea salts
AT-SEA DISPOSAL
Ore slurry transfer with fresh or process water, ship to ship and ship to port
washed ore
PORT
STORAGE PONDS
low-TDS water
electric power
steam
ELUTRIATION PLANT
residual sea salts
DISPOSAL
electric power
reagents
process water
MINERAL BENEFICIATION PLANT
tailings
DISPOSAL
crust concentrate
electric power
coal and coke
slagging agent
sulfiding agent
process water
METALLURGICAL PLANT
steam coal
electric power
liq. petr. gas
sulfur, lime
process water
Ferromanganese and Matte Smelting
Sulfuric Acid Pressure Leach
inert slag
Hi-C FeMn
Co,Ni,Cu matte
Co,Ni,Cu,Zn sulfides
MnO2 (+MnO)
washed tailings (incl. bulk of the Mn)
DISPOSAL OR USE AS AGGREGATE
MARKET
MARKET
DISPOSAL
MARKET, OR REFINING ELSEWHERE
REFINERY IN HAWAII
MARKET, OR REFINING ELSEWHERE
Co and Ni metal mixed sulfides
MARKET

Figure 12. Overview of metallurgical processing.

(From Harvey and Ammann, 1985)

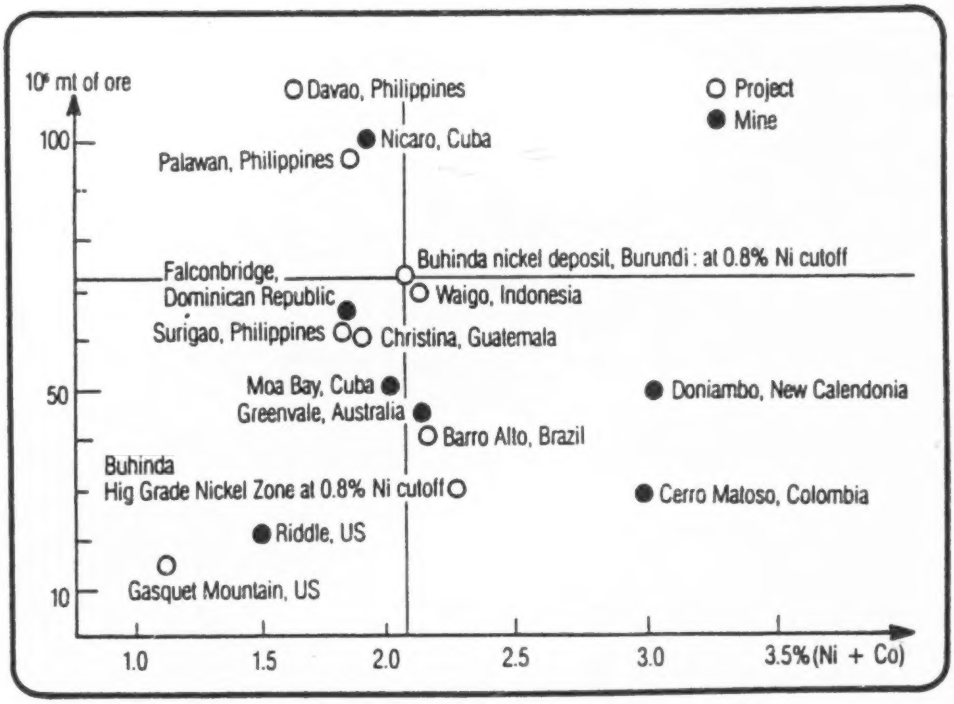


Figure 13. World nickel laterite deposits (E&MJ, July 1986, p.8).

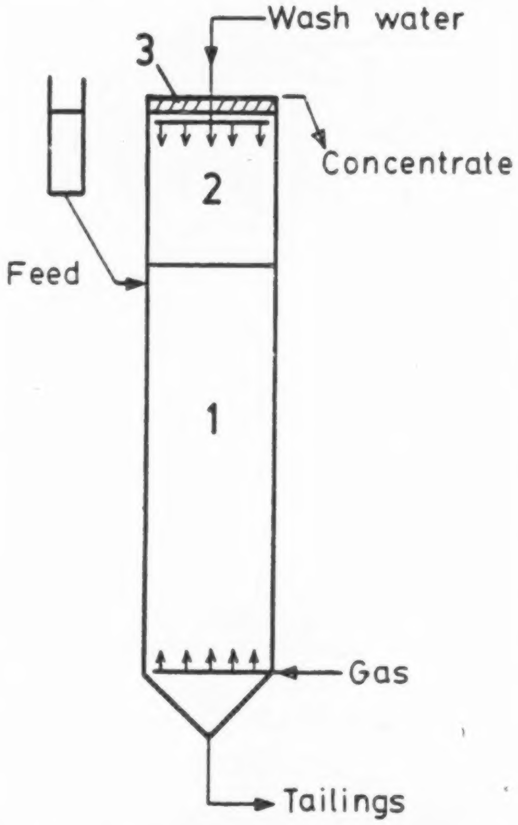

Wash water
3
Concentrate
2
Feed
1
Gas
Tailings

Fig. 14. Schematic illustration of the flotation column: (1) collection zone, (2) washing zone, and (3) thin froth layer. Combined zones 2 and 3 are referred to here as the cleaning zone.

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Appendix

Typical smelter specification for copper concentrates

Payment for copper	Fairly uniform at $(T-1)Q$
Payment for zinc	Never; sometimes penalties
Payment for lead	Never; sometimes penalties
Payment for silver	$0.95TQ$ with minimum deduction of 35 g/t
Payment for gold	$0.95TQ$ with minimum deduction of 1 g/t

Penalties for	For $T >$ %
Zn	2.5
Pb	2.5
Sb	0.1
Bi	0.1
Hg	0.0025
As	0.1

Typical smelter specification for lead concentrates

Payment for lead	Fairly uniform at $0.95TQ$ with minimum deduction of 3% Pb
Payment for zinc	Rare except for ISF-type smelters with minimum deduction between 5 and 10% Pb
Payment for copper	0.9–0.99 paid after 2–2.5% deduction and fairly high refining charge. Another formula quotes payment of $0.2-0.3TQ$
Payment for silver	Fairly uniform at $0.95(T-50)Q$

Penalties for	For $T >$ %	
As	0.1	Very common
Sb	0.1	
Bi	0.01	
S	17	
SiO_2	6	

(From Barbery et al., 1980)

Appendix (Cont'd)

Table 4 Typical smelter specification for zinc concentrates

Payment for zinc	Fairly uniform at 0.85TQ with minimum deduction at 8% Zn
Payment for lead	Rare, except for ISF-type smelters, with minimum deduction of 3 to 8 points %
Payment for copper	Rare
Payment for silver	Fairly uniform and low at 0.60(T−150)Q
Payment for cadmium	Typically 0.6(T−0.2)Q

Penalties for	For $T >$ %	
Bi	0.01	
MgO	0.4	Very common
Hg	0.015	
Sb	0.1	
Cl	0.1	
As	0.1	
SiO_2	6	
Fe	9	

Table 5 Typical smelter specification for lead–copper concentrates

Payment for lead	0.98TQ with minimum deduction of 3% Pb
Payment for zinc	None; penalties
Payment for copper	100%; deduction of 1–3% Cu
Payment for silver	0.95(T−50)Q

Penalties for	For $T >$ %
Zn	10
As	0.1
Bi	0.01
Hg	0.01

(From Barbery et al., 1980)